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TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XXXV.

CONTAINING THE PAPERS AND DISCUSSIONS OF 1904.

NEW YORK CITY:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1905.

WILLIAM HENRY PETTEE.

See Biographical Notice, p. 430.

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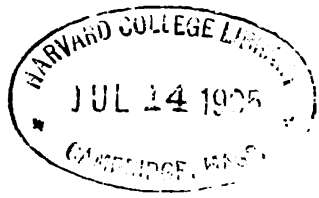

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PREFACE.

THIS volume contains all the published papers and discussions of 1904 (except a few of the latter class, already included in Vol. XXXIV.), together with some discussions, not received for publication until 1905, but relating to papers herein comprised.

Vol. XXXIII. (covering 1902) was delayed until March, 1904, by reason of the extra labor involved in the issue, in 1902, of Vol. XXXII. (the special "Mexican Volume"), as well as Vol. XXXI. (the usual annual publication, covering 1901).

Vol. XXXIV. was, by hard work, distributed to members in November, 1904—a recovery of four months of the time thus lost.

The present volume will be distributed in June—a further gain of five months in date, for the achievement of which I am again indebted to Dr. Joseph Struthers, the efficient Assistant Editor of the *Transactions*.

R. W. RAYMOND.

MAY, 1905.

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OFFICERS.

For year ending February, 1905.

[This list was modified December 29, 1904, by sundry resignations and appointments to fill vacancies thus created, all of which was done to facilitate the legal incorporation of the Institute.]

President.

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(Terms expire February, 1905.)

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(Terms expire February, 1906.)

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(Terms expire February, 1905.)

HEINRICH RIES, B. B. LAWRENCE, F. KLEPETKO.
(Terms expire February, 1906.)

F. LOUIS GRAMMER, CHARLES H. SNOW, JOSEPH HARTSHORNE.
(Terms expire February, 1907.)

Secretary.

R. W. RAYMOND,
99 John St., New York City.

Treasurer.

FRANK LYMAN,
New York City.

The following officers were elected at the Annual Meeting, February, 1905.

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For one year, T. A. Rickard, Charles H. Snow and R. W. Raymond.

For two years, James Gayley, Frank Klepetko and Frank Lyman.

For three years, James Douglas, James F. Kemp and Albert R. Ledoux.

COUNCIL:—James Gayley (*President*), R. W. Raymond (*Secretary*).

VICE-PRESIDENTS:—*For one year*, George W. Maynard, Julian Kennedy and Charles D. Walcott. *For two years*, William P. Blake, Thomas F. Cole and Irving A. Stearns.

COUNCILORS:—*For one year*, Frank Klepetko, Benjamin B. Lawrence and Heinrich Ries. *For two years*, F. L. Grammer, Joseph Hartshorne and Charles H. Snow. *For three years*, A. A. Blow, Frank Lyman and T. A. Rickard.

---

* Deceased, September 12, 1904. The Council appointed T. A. Rickard, New York City, to fill the vacancy for the unexpired term.

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PRESIDENTS.

*DAVID THOMAS	1871
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R. W. RAYMOND	1873
R. W. RAYMOND	1874
*A. L. HOLLEY	1875
*ABRAM S. HEWITT	1876
*T. STERRY HUNT	1877
*ECKLEY B. COXE	1878
*ECKLEY B. COXE	1879
*WILLIAM P. SHINN	1880
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*RICHARD P. ROTHWELL	1882
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JAMES C. BAYLES	1884
JAMES C. BAYLES	1885
ROBERT H. RICHARDS	1886
*THOMAS EGLESTON	1887
WILLIAM B. POTTER	1888
RICHARD PEARCE	1889
*ABRAM S. HEWITT	1890
JOHN BIRKINBINE	1891
JOHN BIRKINBINE	1892
H. M. HOWE	1893
JOHN FRITZ	1894
*J. D. WEEKS	1895
E. G. SPILSBURY	1896
*THOMAS M. DROWN	1897
C. KIRCHHOFF	1898
JAMES DOUGLAS	1899
JAMES DOUGLAS	1900
E. E. OLCOTT	1901
E. E. OLCOTT	1902
ALBERT R. LEDOUX	1903
JAMES GAYLEY	1904

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*THOMAS M. DROWN	1873–1884
R. W. RAYMOND	1884 ——

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*THEODORE D. RAND	1872–1903
FRANK LYMAN	1903

* Deceased.

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M. FLORIS OSMOND.....Paris, France.
PROF. DIMITRY CONSTANTIN TSCHERNOFF.....St. Petersburg, Russia.
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CONTRERAS, MANUEL MARIA.....1902
DAUBRÉE, A.....1896
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GAETZSCHMANN, MORITZ.....1895
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HUNT, T. STERRY.....1892
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ROBERTS-AUSTEN, W. C.....1902
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* Annual meeting for the election of officers.

† Begun in February at New York City, for the election of officers, and adjourned to Florida

‡ " " " " " " " " " " " " " " to Philadelphia

PUBLICATIONS.

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The volumes of *Transactions*, which are published annually, contain the list of officers, rules, etc., the Proceedings, and the papers revised for final publication. (In this revision, after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there are no important changes.) These volumes are for sale as follows, in paper covers:

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Index, Vols. XXVI. to XXX., inclusive, cloth,	1 50
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"*The Genesis of Ore-Deposits*," comprising the famous treatise of the late Professor Franz Posepny, with the successive discussions thereof by Le Conte, Blake, Winchell, Church, Emmons, Becker, Cazin, Rickard and Raymond (all of which were published in Volumes XXIII. and XXIV. of the *Transactions* of the Institute, and subsequently in the special "Posepny Volume," issued by the Institute); also, later papers by Van Hise, Emmons, Weed, Lindgren, Vogt, Kemp, Blake, Rickard and others, and the discussions of these papers by De Launay, Beck, and many others (some of these were included in Volume XXX. and the remainder appeared in Volume XXXI.); also a complete bibliography of the Institute papers and discussions on this subject from 1871 to the present time.

The original Posepny volume comprised 265 pages, and was sold for $2.50, at which price the edition was long since exhausted. The present volume is an octavo of 825 pages, bound in "book-linen," of the same color as the standard binding of the *Transactions*, . . . $6 00

Half-morocco bound copies, ... 7 00

"*The Evolution of Mine-Surveying Instruments.*" This is a volume of about 400 pages, issued in the same style as the foregoing, and containing the original paper of Mr. Dunbar D. Scott on that subject (*Transactions*, XXVIII.), first published in 1898, together with later papers, continuing the same subject, and discussions thereof, by Hoskold, Lyman, Davis and many others, . 3 50

Half-morocco binding, ... 4 50

Glossary of Mining and Metallurgical Terms (1881), cloth, . 50

Spanish-American Mining and Metallurgical Glossary, bound in leather, pocket-size, 96 pages, ... 75

Special Mining and Railway Map of Mexico, size 14 by 20, prepared by order of Dept. of Fomento, 1901, . . . 35

List of Members, Rules, etc., paper, ... 50

Pamphlets.

1. The Minutes of the Proceedings of each Meeting.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. (In nearly all cases in which papers, the titles of which appear in the Proceedings, are not subsequently published, they have been withdrawn by the authors.) These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the edition distributed, without charge, to members and associates not in arrears, a small supply is retained to meet subsequent demand. There are no copies on hand of papers read before 1880. The stock is nearly complete from 1880. These papers are for sale at the office of the Secretary, or are sent to purchasers by mail or express, charges paid, on receipt of the price, as follows:

No. of Pages.	Single Copies.	10 Copies.	20 Copies.
8 or less	$0 10	$0 60	$1 00
9 to 12 inclusive	0 10	0 80	1 40
13 to 16 "	0 12	1 00	1 75
17 to 20 "	0 16	1 25	2 25
21 to 24 "	0 20	1 50	2 75
25 to 40 "	0 25	2 00	3 50
41 to 56 "	0 30	2 50	4 50
57 to 72 "	0 35	3 00	5 00
73 to 88 "	0 40	3 25	5 25
89 to 104 "	0 45	3 50	6 00
105 to 120 "	0 50	3 75	6 25

Papers with folders and inserted plates subject to special price.

Authors' Edition of Pamphlets.

Extra copies, if ordered before the printing of the pamphlet edition for distribution to the members of the Institute, will be furnished to authors, under Rule VII., at special rates, which will be stated on application to the Secretary, R. W. Raymond, 99 John St., New York, N. Y. (P. O. Box 223).

RULES

ADOPTED MAY 1873. AMENDED MAY, 1875, 1877, AND 1878, FEBRUARY, 1880, 1881, 1887, 1890, 1896, 1908 AND 1904.

I.

OBJECTS.

THE objects of the AMERICAN INSTITUTE OF MINING ENGINEERS are to promote the arts and sciences connected with the economical production of the useful minerals and metals, and the welfare of those employed in these industries, by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate, by means of publications among its members and associates, the information thus obtained.

II.

MEMBERSHIP.

The Institute shall consist of Members, Honorary Members, and Associates. Members and Honorary Members shall be professional mining engineers, geologists, metallurgists, or chemists, or persons practically engaged in mining, metallurgy, or metallurgical engineering. Associates shall include all suitable persons desirous of being connected with the Institute, and duly elected as hereinafter provided. Each person desirous of becoming a member or associate shall be proposed by at least three members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) upon receiving three-fourths of the votes cast, and shall become a member or associate on the payment of his first dues. Each person proposed as an honorary member shall be recommended by at least ten members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) on receiving nine-tenths of the votes cast; *Provided*, that the number of honorary members shall not exceed twenty. The Council may at any time change the classification of a person elected as associate, so as to make him a member, or *vice versa*, subject to the approval of the Institute. All members and associates shall be equally entitled to the privileges of membership; *Provided*, that honorary members shall not be entitled to vote, and members or associates whose post-office address shall be outside of the United States, Canada and Mexico shall not be entitled to vote by mail, except upon proposed amendments to the Rules.

Any member or associate may be stricken from the list on recommendation of the Council, by the vote of three-fourths of the members and associates present at any annual meeting, due notice having been mailed in writing by the Secretary to the said member or associate.

III.

DUES.

The dues of members and associates shall be $10 each per annum, payable in advance on the first day of each calendar year; and persons elected at any time during any calendar year shall pay the dues of that year upon election. And, in addition to the above dues, the Council may at any time prescribe an initiation-fee of $10, for all candidates thereafter proposed.* Any member or associate not in arrears may become, by the payment of one hundred and fifty dollars at one time, a life-member or associate, and shall not be liable thereafter to annual dues. Any member or associate in arrears may, at the discretion of the Council, be deprived of the receipt of publications, or stricken from the list of members when in arrears for one year; *Provided*, that he may be restored to membership by the Council on payment of all arrears, or by re-election after an interval of three years.

* NOTE.—This provision became operative, April 18, 1904.

IV.

OFFICERS.

The affairs of the Institute shall be managed by a Council, consisting of a President, six Vice-Presidents, nine Managers, a Secretary and a Treasurer, who shall be elected from among the members and associates of the Institute at the annual meetings, to hold office as follows:

The President, the Secretary, and the Treasurer for one year (and no person shall be eligible for immediate re-election as President who shall have held that office subsequent to the adoption of these rules, for two consecutive years), the Vice-Presidents for two years, and the Managers for three years; and no Vice-President or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. At each annual meeting a President, three Vice-Presidents, three Managers, a Secretary, and a Treasurer shall be elected, and the term of office shall continue until the adjournment of the meeting at which their successors are elected.

The duties of all officers shall be such as usually pertain to their offices, or may be delegated to them by the Council or the Institute; and the Council may in its discretion require bonds to be given by the Treasurer. At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Vacancies in the Council may occur by death or resignation; or the Council may, by a vote of the majority of all its members, declare the place of any officer vacant, on his failure for one year, from inability or otherwise, to attend the Council meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *Provided*, that the said appointment shall not render him ineligible at the next annual meeting.

Five members of the Council shall constitute a quorum; but the Council may

appoint an Executive Committee, or business may be transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to the approval of a majority of the Council, subsequently given in writing to the Secretary, and recorded by him with the minutes. The Council may at any time appoint a trustee or trustees, to hold property, real or personal, for the use and benefit of the Institute, upon such terms and conditions as the Council may determine.

V.

ELECTIONS.

The annual election shall be conducted as follows: Nominations may be sent in writing to the Secretary, accompanied with the names of the proposers, at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before the said meeting, mail to every member or associate (except honorary members) a list of all the nominations for each office so received, together with a copy of this rule, and the names of the persons ineligible for election to each office; and if the Council, or a Committee thereof, appointed for the purpose, shall have recommended any nominations, such recommendation may also be sent to members and associates with the said list of all nominations made, but not upon the same paper. And each member or associate, qualified to vote, may vote, either by striking from or adding to the names of the said list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing said altered or prepared ballot with his name, and either mailing it to the Secretary or presenting it in person at the annual meeting; *Provided*, that no member or associate in arrears since the last annual meeting shall be allowed to vote until the said arrears shall have been paid. The ballots shall be received and examined by three Scrutineers, appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected, and the Scrutineers shall so report to the presiding officer. The ballots shall be destroyed, and a list of the elected officers, certified by the Scrutineers, shall be preserved by the Secretary.

VI.

MEETINGS.

The annual meeting of the Institute shall take place on the third Tuesday of February, at which a report of the proceedings of the Institute and an abstract of the accounts shall be furnished by the Council. Other meetings shall be held in each year, at such times and places as the Council shall select, and notice of all meetings shall be given by mail, or otherwise, to all members and associates, at least twenty days in advance.

Every question, which shall come before any meeting of the Institute, shall be decided, unless otherwise provided by these Rules, by the votes of a majority of the members then present. Any member or associate may introduce a stranger to any meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

VII.

PAPERS AND PUBLICATIONS.

The Council shall have power to decide on the propriety of communicating to the Institute any papers which may be received, and they shall be at liberty, when they think it desirable, to direct that any paper read before the Institute shall be printed in the *Transactions*. Intimation, when practical, shall be given, at each general meeting, of the subject of the paper or papers to be read, and of the questions for discussion at the next meeting. The reading of papers shall not be delayed beyond such hour as the presiding officer shall think proper; and the election of members or other business may be adjourned by the presiding officer, to permit the reading and discussion of papers. The published papers and volumes of *Transactions* shall be distributed to all members and associates not in arrears, and may be sold to the public upon such conditions as the Council shall prescribe; but the Council may, in its discretion, omit sending to members and associates outside of the United States, Canada and Mexico, special circulars, unless the same contain proposed amendments to the Rules.

The copyright of all papers communicated to, and accepted by, the Institute, shall be vested in it, unless otherwise agreed between the Council and the author. The author of each paper read before the Institute shall be entitled to twelve copies, if printed, for his own use, and shall have the right to order any number of copies at the cost of paper and printing, provided said copies are not intended for sale. The Institute is not, as a body, responsible for the statements of fact or opinion advanced in papers or discussions at its meetings, and it is understood that papers and discussions should not include matters relating to politics or purely to trade; nor shall the Council or the Institute officially approve or disapprove any technical or scientific opinion or any proposed enterprise outside the management of the meetings, discussions and publications of the Institute, as provided in these Rules; *Provided*, however, that committees may be appointed by the Council or the Institute to make investigations and submit reports at meetings of the Institute; but no action shall be taken binding the Institute for or against the conclusions of any such reports.

VIII.

AMENDMENTS.

These Rules may be amended at any annual meeting by a two-thirds vote of the members present; *Provided*, that written notice of the proposed amendment shall have been given at a previous meeting; *and Provided, also*, that the amendment or amendments so adopted shall be printed upon a ballot and sent, not later than the next distribution of printed matter, to all members and associates not in arrears for the preceding year (except honorary members and foreign members elected before February, 1880), and each person receiving the same shall be requested to return it to the Secretary with his written vote of Yes or No to each amendment, and his signature; and the President shall appoint as Scrutineers three members or associates, who shall examine all of the said ballots which shall have been returned within one month from the date of their distribution, and shall report the result; and the Secretary shall publish and distribute to members, not later than the next distribution of printed matter, an announcement of the said result so reported, together with the text of the additional or amended rule or rules so

adopted; and the amendment or amendments approved by the majority of the ballots so returned and reported shall become part of these Rules from and after the publication of said announcement by the Secretary.

IX.

INCORPORATION.

The Council is hereby authorized to cause to be created under the statutes of the State of New York, a corporation to hold and administer, for the use and benefit of the Institute, such real or personal property as the Council may, from time to time, transfer to it, such transfer being hereby authorized; said corporation not to be subject to control by the members and associates of the Institute, except through the Council, and in such way as may be permitted by the articles of its incorporation, approved by the Council. *Provided*, that the regular business of the Institute, namely, the holding of meetings, and the preparation and distribution of publications, shall remain completely in the control of the Council; and that the funds of the Institute, derived from the fees and dues of members and associates or the sale of publications, shall not be transferred to, or held, or controlled by, such corporation, except so far as the Council of the Institute may appropriate money for the necessary expenses of the same—the purpose of this provision being, that the regular business of the Institute, as above defined, shall remain in the hands of the Council, without interference on the part of the said corporation.

And the Council is hereby authorized to give its assent and co-operation to the generous offer of Andrew Carnegie, a member of this Institute, to provide a building in the City of New York for the use of American Engineering Societies; and in case a corporation should be formed to own and administer such building, the Council is authorized to name representatives for the Institute in the managing board of said corporation.

Proceedings of the Eighty-Sixth (Thirty-Fourth Annual) Meeting, Atlantic City, N. J., February, 1904.

THE City of Baltimore, Md., was originally selected for the place of the annual meeting; but the disastrous conflagration which began there on February 7, and devastated a large part of the business section of the city, necessitated the immediate selection of another place of meeting. The choice of Atlantic City, N. J., at so late a date, left no time for the arrangement of excursions or entertainments, such as have always formed an attractive feature of the meetings of the Institute. The hotel-headquarters, bureau of information, and hall for all the sessions, were in the Hotel Rudolf.

The opening session was held on Tuesday evening, February 16th. President Albert R. Ledoux called the meeting to order and welcomed the members of the Institute and their guests. The following letter from the Honorable Edwin Warfield, Governor of the State of Maryland, was read:

EXECUTIVE DEPARTMENT.

ANNAPOLIS, MARYLAND.

BALTIMORE, MD., February 12, 1904.

J. H. LEE, ESQ., 1901 N. Charles St., Baltimore, Md.

My Dear Sir: The Governor directs me to acknowledge receipt of your letter of the 10th instant, informing him of the change of plans of the American Institute of Mining Engineers in view of the great calamity that has overtaken our city. He regrets that he will thus lose the opportunity of meeting your members, to which he had been looking forward with much pleasure.

Very truly yours,
(Signed) R. S. HART,
Secretary to the Governor.

Letters were read from Mr. J. Henry Lee, Chairman, and Mr. William Glenn, of the Local Committee of the proposed Baltimore meeting; and the welcome announcement was made that the Institute flag, which has graced every meeting of the Institute for many years, was safe, and would be presented by Mr. Lee on the morrow.

President Ledoux then delivered the Presidential address, The American Mining Engineer.*

Mr. Harrison Souder gave an oral abstract of his paper, Mineral Resources of Santiago, Cuba, which was followed by an interesting discussion.

The following paper was presented in oral abstract:

Biographical Notice of Robert Henry Thurston, by R. W. Raymond, New York City.

The President announced the appointment, as Scrutineers to examine the ballots received and report the names of the officers elected for the ensuing year, of Messrs. B. F. Fackenthal, Jr., Edgar S. Cook and Walter Wood.

The second session, held on Wednesday morning, February 17, was devoted to the reading and discussion of the papers on iron and steel.

The following papers were presented by the authors:

Specifications for Cast-Iron and Finished Castings, by Richard Moldenke, New York City.

Chemical Specifications for Pig-Iron, by Edgar S. Cook, Pottstown, Pa.

The Need of Standard Specifications for Gray-Iron Castings, by Henry Souther, Hartford, Conn.

The following papers were presented in printed form:

Standard Specifications for Pig-Iron and Iron Products, by a Sub-Committee of the American Society for Testing Materials.

Note on the Further Discussion of the Physics of Cast-Iron, by William R. Webster, Philadelphia, Pa.

Notes on the Physics of Cast-Iron, by Richard Moldenke, New York City.

Specifications for Pig-Iron and Iron Castings, by Robert Job, Reading, Pa.

The Standardization of Specifications for Iron and Steel: Recent Progress in America and England, by William R. Webster and Edgar Marburg, Philadelphia, Pa.

Notes on Rail-Steel, by Robert W. Hunt, Chicago, Ill.

A Decade in American Blast-Furnace Practice, by F. Louis Grammer, Baltimore, Md.

* Published and distributed in pamphlet form, but not included in this volume.

Direct-Metal and Cupola-Metal Iron Castings, by Thomas D. West, Sharpsville, Pa.

Stock-Distribution and its Relation to the Life of a Blast-Furnace Lining, by David Baker, Sydney, Cape Breton, Canada.

The following papers were read by title for subsequent publication:

Standard Specifications for Locomotive Cylinders, by Walter Wood, Philadelphia, Pa.

Specifications for Cast-Iron, by C. R. Baird, Philadelphia, Pa.*

Standard Specifications for Cast-Iron Car-Wheels, by Charles B. Dudley, Altoona, Pa.

Specifications for Malleable Cast-Iron, by Stanley G. Flagg, Jr., Philadelphia, Pa.*

The third session was held on Wednesday afternoon, February 17th.

The following papers were presented in oral abstract by the authors:

Standard Specifications for Cast-Iron Pipe, by Walter Wood, Philadelphia, Pa.

The Mobility of Molecules of Cast-Iron, by A. E. Outerbridge, Jr., Philadelphia, Pa. (This paper was illustrated by several interesting specimens of cast-iron, which had been increased in volume up to 46 per cent. by the treatment described.)

The following papers were read by the Secretary in the absence of the authors:

Geogenesis and Some of Its Bearings on Economic Geology, by Persifor Frazer, Philadelphia, Pa.

Radium and Radio-Active Minerals, and the Search for Them, by George F. Kunz, New York City.†

The fourth and final session was held on Thursday morning, February 18th.

The following papers were presented by the authors:

The Use of High Percentages of Mesabi Iron-Ores in Coke Blast-Furnace Practice, by W. A. Barrows, Jr., Sharpsville, Pa.

The Separation and Concentration of Graphite by Oil, by F. M. Zeller, Philadelphia, Pa.*

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* Not furnished for publication.

† Published only in preliminary pamphlet form.

The following papers were presented in printed form:

A Bituminous-Coal Breaker, by Lewis Stockett, Stockton, Mont.

Fuel and Mineral Briquetting, by Robert Schorr, San Francisco, Cal.

Concrete in Mining and Metallurgical Engineering, by Henry W. Edwards, Grand Junction, Col.

Wet Methods of Extracting Copper at Rio Tinto, Spain, by Charles H. Jones, Santiago, Cuba.

Note on the Relation between Arsenic and Electro-Motive Force in Copper-Electrolysis, by L. Webster Wickes, Anaconda, Mont.

Notes upon Preliminary Tests and Cyanide-Treatment of Silver-Ores in Mexico by the MacArthur-Forrest Process, by John F. Allan, Mexico City, Mexico.

The Plotting of Sizing-Tests, by W. Spencer Hutchinson, Boston, Mass.

Note on the Cost and Speed of Sinking the East Shaft of the New Kleinfontein Co., Benoni, South Africa, by Edward J. Way, Benoni, Transvaal, South Africa.

Estimated Costs of Mining and Coking and Relative Commercial Returns from Operating in the Connellsville and Walston-Reynoldsville Districts, Pa., by Edward V. d'Invilliers, Philadelphia, Pa.

Discussion of the Paper of Mr. Watson, Geological Relations of the Manganese-Ore Deposits of Georgia, by Charles Catlett, Staunton, Va.*

Discussion of the Paper of Mr. Merrill, The Metallurgy of the Homestake Ore, by Mark R. Lamb, Sonora, Mexico.*

Discussion of the Paper of Waldemar Lindgren, The Geological Features of the Gold Production of North America, by H. W. Turner, San Francisco, Cal.*

The following papers were read by title for subsequent publication and discussion:

Notes on the Gold District of Canutillo, Chile, S. A., by Sydney H. Loram, Canutillo, Chile, S. A.

Testing Gold-Ores by Amalgamation, by Ernest A. Hersam, Berkeley, Cal.

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* Published in *Trans.*, xxxiv., because the original paper is contained in that volume.

Sulphide-Smelting at the National Smelter of the Horseshoe Mining Co., Rapid City, S. D., by Charles H. Fulton and Theodor Knutzen, Rapid City, S. Dak.

Additional Remarks on Surveying-Instruments, by H. D. Hoskold, Buenos Ayres, So. Amer.

Origin of the Magnetic Iron-Ores of Iron County, Utah, by E. P. Jennings, Oak Park, Ill.

The Equipment of a Laboratory for Metallurgical Chemistry in a Technical School, by Charles H. White, Boston, Mass.

Notes and Observations on Cast-Iron, by J. E. Johnson, Jr.

Discussion of the Paper of R. W. Lodge, The Assay of Zinc-Box Residues, by Charles H. Fulton, Rapid City, S. D.*

Discussion of the Paper of W. Randolph Van Liew, Relative Elimination of Impurities in Bessemerizing Copper-Matte, by Allan Gibb, London, England.*

The Volcanic Origin of Oil,† by Eugene Coste, Toronto, Canada.

The Annual Report of the Council was read as follows:

Annual Report of the Council.

In accordance with the Rules, the Council makes the following report to the Institute for the year ending December 31, 1903:

The Financial Statement of the Secretary and the Treasurer show receipts from all sources for the year ending December 31, 1903 (including the balance of $11,574.67 on hand December 31, 1902), of $56,536.99, and expenditures of $44,443.84 (including the purchase for $5,467.50 of $6,000 New York Central, Lake Shore Trust Collateral 3.5 per cent. bonds), leaving $6,625.65 cash on hand. No account is taken in this statement of the increased value of the assets of the Institute, in back volumes of the *Transactions*, office furniture, etc. In addition to the cash on hand, the Institute possesses invested funds of the par value of $21,900, and market value of more than $24,000, yielding about $1,400 interest annually; and on December 31, there were no bills payable for outstanding obligations. The detailed statement is as follows:

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* Published in *Trans.*, xxxiv., because the original paper is contained in that volume.

† Printed partly in discussion of the paper by Mr. R. T. Hill, The Beaumont Oil-Field, with Notes on Other Oil-Fields in the Texas Region, *Trans.*, xxxiii., 363.

Receipts.

Balance from statement of December 31, 1902,		$11,574 67
Annual dues,	$31,624 98	
Life memberships,	4,145 74	
Binding of *Transactions*,	4,067 80	
Sale of publications,	3,526 43	
Electrotypes,	38 50	
Interest on bonds and deposits,	1,478 16	
Miscellaneous,	80 71	44,962 32
		$56,536 99

Disbursements.

Printing volume xxxii. of the *Transactions*,	$2,975 34	
" volume xxxiii. " " on account,	2,218 60	
" pamphlet edition of papers,	5,446 98	
" new volume of Ore-Deposits,	177 79	
" circulars and ballots,	648 19	
Binding volume xxxii. and miscellaneous volumes,	2,883 70	
" exchanges,	153 55	
Engraving and electrotyping,	1,109 80	
Secretary's department, including clerks, stenographers, and expenses of editing and proof-reading,	9,890 00	
Postage, including post-office box-rent,	3,899 42	
Stationery,	844 41	
Rent,	2,500 00	
Express- and freight-charges,	1,963 97	
Telephone,	193 50	
Telegrams, cablegrams and car-fares,	105 44	
Office-equipment,	321 93	
Assistant Treasurer's department,	4,904 00	
Storage of *Transactions*,	188 42	
Special stenographers and expense of meetings,	360 11	
Office-supplies and repairs,	512 26	
Refunding payments,	1 73	
Insurance,	36 10	
Collection-charges,	25 98	
Extra clerical help,	516 58	
Library-additions,	859 52	
Librarian and assistant (salaries),	1,027 60	
Translations,	20 00	
Special "onion-skin" paper for illustrations,	639 36	
Miscellaneous,	19 56	44,443 84
Purchase of N. Y. C. & H. R. R. R. Lake Shore Trust Collateral bonds,		5,467 50
Balance in cash account,		6,625 65
		$56,536 99

It will be seen that this statement indicates that the income and expenditure of the year have been nearly equal, the cash

on hand December 31, 1903, being but $518.48 more than the amount on hand December 31, 1902, less the amount invested in the purchase of bonds. The expenditures of the year, however, have been unusually large, on account of the publication of an extra volume of the *Transactions* (vol. xxxii., the "Mexican Volume"), which has been sent to members without extra charge. Not only the cost of printing and distributing this volume, but also that of translating, engraving and editing it, and of the extensive correspondence connected therewith, have largely increased the expenditures of the past year, though, having been partly paid for in 1902, they are not wholly included in the above statement.

Another result of this extra labor and expense has been the delay in issuing Vol. xxxiii. of the *Transactions*, which is, however, now in the binders' hands, and which is not inferior, in size or professional value to its predecessors.

Two additional items of expenditure in the forgoing statement deserve mention, namely, the cost of library additions and the salary of a librarian and assistant. With regard to these items, the following statement is appropriate:

After the death of Mr. R. P. Rothwell, one of the founders and at a later date, President, of the Institue, his private library of bokos on engineering was purchased by Mr. John Hays Hammond, and presented to the Institute in memory of Mr. Alfred Raymond, the son and (at the time of his death) editorial assistant of the Secretary.

After the death of Mr. Clarence King, his private library of books on geology, etc., was similarly purchased and presented by Mr. Hammond.

To these generous contributions Dr. R. W. Raymond, Secretary of the Institute, added the gift of his own private library of books on mining, metallurgy, etc.

These three collections were added to the collections of books of reference, sets of periodicals, etc., already in the Secretary's office, the separate volumes being simply marked by a bookplate as presented in memory of Alfred Raymond.

Subsequently, upon the death of Mr. William Van Slooten, a member of the Institute, his widow presented to the Institute his private library of works on engineering.

It is interesting to note that, among the thousands of volumes

thus donated from different sources, there were only a couple of hundred duplicates. Mr. Rothwell's library was rich in French, Dr. Raymond's in German, and Mr. Van Slooten's in English and American, authorities; while that of Mr. King was extremely valuable in the sphere of geology, rather than engineering proper.

No official notice of these donations was given by the Council, because the Institute, as an unincorporated body, was not prepared to accept the ownership of property so valuable. The books are therefore temporarily held by trustees, until the Institute shall have authorized the incorporation of a suitable body to hold and administer the Institute library. An amendment to the Rules, proposed at the New York meeting of October last, and to be considered at the annual meeting of February, 1904, will, if adopted, authorize this step.

It is hoped that the thorough indexing, preservation and increase of this library will be provided for hereafter by special contributions to a fund set apart for that purpose. In fact, a small sum has been already contributed to the present Trustees for that purpose, but this has not been drawn upon for the expenses of the library, the whole of which have been paid, thus far, from the ordinary income of the Institute.

The expenditures under this head during the year have been, as shown in the above statement, for binding, purchase of volumes and salaries of librarian and assistant, about $2,000, not including the proportion of rent chargeable to the rooms occupied by the library.

Including the former library of exchanges, books of reference, etc., the collection now comprises:

Bound volumes,	9,500
Unbound volumes,	2,500
Maps,	500
Pamphlets,	2,750
Total,	15,250

besides about 450 duplicates, available for sale or exchange.

Through the skill and devotion of Miss L. E. Howard, the efficient librarian, a complete card-catalogue for the use of members has been prepared and kept up to date, and a pamphlet catalogue of 43 pages, giving a list of periodicals on file,

etc., will be issued at an early date. Several hundred members and others have availed themselves already of the opportunity to consult this library, which is believed to be, as a record of modern progress and practice within its sphere, second to none in the United States.

Two meetings were held during the year: The annual meeting in February, at Albany, N. Y., and a meeting in October, at New York City. The official reports of the proceedings, excursions, etc., of these meetings, and the pages of our *Transactions* containing their papers and discussions, are sufficient evidence of their social interest and professional value.

The most important event of the year, in its bearing upon the future history of the Institute, was the proposal by Mr. Andrew Carnegie, a member of many years' standing, to erect a suitable building for the use of national American engineering societies. The following statement of the proceedings of the Council with reference to this plan, although already made known by circular, is here reprinted, in order to place it permanently on record in the *Transactions:*

In February, 1903, Mr. Carnegie wrote the following letter:

"NEW YORK, February 14, 1903.

"*Gentlemen American Society Civil Engineers, American Society Mechanical Engineers, American Institute Mining Engineers, American Institute Electrical Engineers and the Engineers' Club:*

"It will give me great pleasure to give, say, one million dollars to erect a suitable union building for you all, as the same may be needed.

"With best wishes, truly yours,

"ANDREW CARNEGIE."

In addition to this munificent gift (afterwards increased to $1,500,000), Mr. Carnegie undertook to advance the cost of the land required, and has already paid for that purpose a large sum.

After repeated informal conferences of the members of the governing bodies of the several organizations named, and of committees appointed by said bodies, the following preamble and resolutions were unanimously adopted by the Council of the Institute, June 4, 1903:

Whereas, Mr. Andrew Carnegie, a member of the Institute, has offered the sum of about $1,000,000 to erect a suitable building for the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute

of Electrical Engineers, the American Institute of Mining Engineers, and the Engineers' Club of New York City;

Now, therefore, resolved, That the Council of this Institute extends to Mr. Carnegie a cordial and grateful recognition of his generous offer, and will do all in its power to secure the attainment of his wise and beneficent purpose and the participation of this Institute therein;

And whereas, it is understood that the said offer of Mr. Carnegie, though made to all the organizations above named, will hold good for any which may accept it, and it is desirable to ascertain, at the earliest possible time, which of the said organizations will unite in accepting it;

And whereas, this Institute, as such, has not the power to hold real estate (except that real or personal property can be held by a trustee for its use and for the purposes which the Institute is organized to promote) and could not be incorporated without very considerable delay, and, even if it had or should acquire such power, it would be inadvisable that it should become tenant in common with other organizations of any real estate, by reason of the complications which might ensue upon the dissolution or insolvency of any one thereof;

Now, therefore, resolved, That the American Institute of Mining Engineers will unite with the other organizations above named, or any two of them, to accept the offer of Mr. Carnegie, to the full extent to which the same can now be done by this Council.

Resolved, That upon the completion of the proposed building upon the site on the north side of 39th Street the offices of this Institute be removed to such building; and the Institute will accept and occupy in said building such space as may be assigned to it by a joint Committee, which shall consist of an equal number of representatives from each engineering society participating in the said gift of Mr. Carnegie, and will pay, in lieu of rent, its share of such annual expense for interest on the cost of the land, maintenance, repairs, heating and lighting, etc., of said building as may be reasonably assessed upon it by the said joint Committee or any equivalent managing board hereafter created, with the approval of the said representatives of the Institute, which share shall be based on the amount of space assigned to the Institute as compared with that assigned to the other organizations.

Resolved, That the title to the said building shall not be held by the organizations uniting in the acceptance of Mr. Carnegie's gift as tenants in common, but shall be held by another corporation, or by trustees, in such manner that the building may continue perpetually to serve the purpose of its munificent founder.

Resolved, That for the purpose of ascertaining and determining the details mentioned in the resolutions already adopted by this Council, and for all further conference on said details with the representatives of the other organizations concerned, the present Conference Committee, heretofore appointed by this Council, consisting of Messrs. A. R. Ledoux, Charles Kirchhoff and Theodore Dwight, is hereby continued and authorized to represent this Council in all matters within its power on this subject; and the said A. R. Ledoux, Charles Kirchhoff and Theodore Dwight are hereby authorized to act as the representatives of this Institute as members of any joint committee, board of trustees, incorporated or otherwise, that shall have charge of the design, erection and maintenance of said building, under such rules and restrictions as may be determined by the Deed of Trust, the said Trustees, or the joint committee of management that may be appointed by the engineering societies which are beneficiaries under this trust; *Provided*, that the Council may at any time appoint a substitute for any member of said committee, or discharge said committee and appoint a new one.

Resolved, That the Secretary is authorized to append to the minutes of this meeting such statement as may be prepared on this subject by the individual members of this Council present at this meeting; to send the same for individual signature to all members of Council not present at this meeting and record with the minutes of this meeting the names of those who sign the same; and subsequently to submit the same by circular to the members and associates of the Institute, and to invite their opinion thereon.

The following is the individual statement above mentioned:

NEW YORK CITY, June 10, 1903.

To The Members and Associates of the American Institute of Mining Engineers:

In accordance with a resolution of Council adopted June 4, 1903 (a copy of which, together with other resolutions of the same date, is forwarded herewith), the undersigned, members of the Council, make as individuals the following statement and inquiry:

1. We unanimously favor the co-operation of the Institute with the other organizations named in accepting the generous offer of Mr. Carnegie.

2. It may be necessary for the full accomplishment of that purpose to amend the rules of the Institute so as to permit it to be represented jointly with other organizations in the administration of Mr. Carnegie's gift. Such amendment must be proposed in writing at the October meeting, adopted at the Annual Meeting of February, 1904, and subsequently by ballot through the mail, as prescribed in Rule VIII.

3. It is necessary that the practical co-operation of the Institute in the proposed plan shall be pledged without such delay. The Council has done all that it can legally do towards such a pledge; and it is understood that the personal assurance of the members of the Council as to further necessary measures will suffice for the present to make the co-operation of the Institute effective in the acceptance of Mr. Carnegie's gift. The undersigned are ready to make such a declaration if assured of the support of the membership of the Institute. The replies received to this statement will not constitute formal action on the part of the Institute, but, if substantially unanimous and favorable, will warrant the undersigned in assuming the personal responsibility of promising such future action as may be necessary.

4. The following information is desired from each member or associate:

a. Do you favor the co-operation of the Institute in the acceptance of Mr. Carnegie's gift?

b. Would you vote to amend the Rules of the Institute in such a way as to permit the incorporation of a smaller body, to hold its property, and be its legal representative? This plan is strongly favored by the undersigned.

c. Would you vote for an amendment to the Rules, authorizing the Council to appoint a Trustee or Trustees, to hold in trust for the Institute any lease or title necessary to be thus taken or held?

A. R. LEDOUX, S. F. EMMONS, JAMES GAYLEY, J. HENRY LEE, JOHN MARKLE, PHILIP W. MOEN, JAMES F. KEMP, GEORGE A. CROCKER, HORACE V. WINCHELL, CLEMENS C. JONES, E. W. PARKER, JAMES W. NEILL, M. D. VALENTINE, HEINRICH RIES, B. B. LAWRENCE, F. KLEPETKO, FRANK LYMAN, R. W. RAYMOND.

The reply to this statement was very general, and more than 99.25 per cent. of the vote was unqualifiedly favorable under each head.

In consequence of this practically unanimous assurance of support, the corresponding amendments to the Rules were proposed at the October meeting in New York City, to be discussed and approved at the annual meeting of February, and subsequently submitted to the Institute for adoption by postal ballot. Meanwhile, the conference-committee already constituted, and consisting of Messrs. Ledoux, Kirchhoff and Dwight, was authorized to pledge the co-operation of the Institute, and to represent it in further consultation or action with other societies. The following report from that committee exhibits the situation of affairs at the end of 1903:

Report.

The general committee of fifteen—five representatives from each of the beneficiaries under Mr. Carnegie's proposition—has had numerous meetings during the past year. Sub-committees have been formed for handling certain details; one to study carefully the question, how the building is to be held for the benefit of the four engineering societies; the Club building being kept separate, designed, built and maintained by the Club alone, without interference on the part of any of the engineering societies. Another sub-committee has had charge of the question of employing architects and designing the engineering building.

Upon its being demonstrated to Mr. Carnegie that the Civil Engineers were hesitant about joining us, for fear that they would not have adequate quarters, but that this difficulty could be overcome if the building were larger, Mr. Carnegie in writing obligated himself to increase his gift, previously stated at one million dollars, to one million five hundred thousand dollars. Upon its being further shown to Mr. Carnegie that the Civil Engineers feared lest tall buildings on either side of the union engineering building might cut off its light, he voluntarily offered to buy adjoining property, say fifty feet on each side, and put upon it a perpetual restriction that no structure should be erected upon it exceeding five stories in height.

The four engineering societies have each taken such steps as were necessary to enable them to accept the gift and to occupy their respective portion of the building when erected, excepting only the Society of Civil Engineers, which has put the question to a postal-card vote of its members, which vote will be received and counted during the first week in March.

Mr. Carnegie has intimated very strongly that the object of his benefaction was to demonstrate to the world the spirit of reciprocity and co-operation which pervaded American technical men, a spirit which he found lacking in Great Britain, where, as he put it, "each engineer is apt to be an island."

There has been some misapprehension lest what was intended by the occupancy of a union building was some sort of federation of engineering societies whereby each would lose its identity. Nothing of this kind has been thought of or suggested, so far as I am aware. While each society will contribute its share of the expenses and receive *pro rata*—upon some proper basis—its share of revenue, if any there be, from the building, each will preserve its integrity, occupying separate floors of the building, having separate library, alcoves and meeting-rooms, but having the right, under equitable provisions, to make use of the large general auditorium and the large general library.

The Committee on Organization proposes that a special Act, incorporating a committee to administer the engineering building shall be obtained from the New York State Legislature. This committee to consist of, say, four members, one each from the participating engineering organizations.

At this time everything is waiting for the decision of the Civil Engineers, for Mr. Carnegie, having done all that was possible to induce all the societies to join, has intimated a strong probability that he would decide to withdraw his offer if any one of the societies should decide to stay out.

ALBERT R. LEDOUX,
Chairman of the Carnegie Conference Committee of the Institute.

Changes in membership have taken place during the year as follows: 358 members and 57 associates have been elected; 11 associates have become members; 4 members have been rein stated in payment of arrears of dues, coupled with satisfactory explanation of previous default; 1 honorary member, 53 members and 5 associates have died; 46 members and 3 associates have resigned; and 66 members and 5 associates have been dropped for non-payment of dues. These changes are tabulated as follows:

Membership of the American Institute of Mining Engineers, January 1, 1904.

	H. M.	M.	A.	Totals.
Membership Jan. 1, 1903	10	3,100	157	3,267
Gains: By Election	.....	358	57	415
Change of Status	.....	11	.....	11
Reinstatement	.....	4	.....	4
Losses: By Resignation	.....	46	3	49
Dropping	.....	66	5	71
Change of Status	.....	.....	11	11
Death	1	53	5	59
Total gains	.....	373	57	430
Total losses	1	165	24	190
Membership Jan. 1, 1904	9	3,308	190	3,507

The list of deaths, reported during the year 1903, comprises the following names:*

Honorary Member.—J. Peter Lesley (1890).

Members and Associates.—Franklin Ballou (1900); J. C. Bartlett (1877); John F. Blandy (1871); Henry M. Boies (1887); Edwin A. Cade (1903); Prof. Albert H. Chester (1871); Joseph K. Clark (1887); Victor M. Clement (1887); F. M. Coghlan (1891); Henry Coom (1902); Louis Davidson (1899); C. M.

* The figures in parentheses indicate the year in which the persons named were elected by the Institute.

Dickerson (1902); Ralph Dillon (1902); F. C. Dobler (1903); William E. Dodge (1900); W. C. Freeman (1877); William Garrett (1881); Dana Harmon (1900); Otto Heckelmann (1902); Richard P. Heckscher (1893); Abram S. Hewitt (1871); Frank A. Holdsworth (1899); William Hooper (1878); Thomas Jenkins (1901); B. F. Jones (1878); Julian A. Kebler (1885); Fletcher H. Knight (1890); Wilbur C. Knight (1900); Robert P. Lindermann (1890); J. S. Luckraft (1894); E. V. McCandless (1879); George S. Morison (1879); B. F. Morley (1895); H. L. Moulder (1899); F. C. Osgood (1887); John H. Paddock (1882); A. E. W. Painter (1873); James F. Parks (1900); C. F. von Petersdorff (1902); A. G. Phillips (1891); James Henry Pomeroy (1899); J. H. Powell, (1900); E. A. Quintard (1889); Theodore D. Rand (1873); Jacob M. Rich (1883); John T. Richards (1884); Pedro P. Rioseco (1898); August Sahlberg (1900); Dr. Charles Schäffer (1892); George McL. Spotswood (1897); Otis J. Stantial (1900); Dr. Oscar Szontagh (1889); George R. Taylor (1892); W. J. Taylor (1875); Prof. Robert H. Thurston (1875); Robert B. Turner (1895); Frederick A. Wright (1902).

Of these, Messrs. Lesley, Hewitt, Rand, Dodge, Thurston and Blandy have been made the subjects of special Biographical Notices, to be separately published. Concerning the remainder of the list, the data, comprising what the Secretary has been able to obtain, has been published in *Bi-Monthly Bulletin*, No. 1, January, 1905, and sent to all members, subscribers and libraries. The *Bulletin*, being bound, affords a more lasting book of reference than the former pamphlet edition of papers. The usual custom of reprinting these biographical notices in the bound volume of the *Transactions* has not been followed this year for the reason that it is already too large and bulky for convenience and safety. Moreover, in order to keep the size of this and of future volumes within a desirable limit of weight, and yet not exclude papers of live technical interest, it is under consideration to publish a supplementary volume every five years which will contain the biographical notices of all members that have died during that period.

The following amendments to the Rules, duly announced at the New York meeting, October, 1903, were discussed and,

after amendment, were adopted in the following form, subject to final adoption by postal ballot of the members and associates of the Institute.

I.

The first two sentences of Rule III. are amended to read as follows:

"The dues of members and associates shall be $10 each per annum, payable in advance on the first day of each calendar year; and persons elected at any time during any calendar year shall pay the dues of that year upon election. And, in addition to the above dues, the Council may at any time prescribe an initiation-fee of $10, for all candidates thereafter proposed."

II.

Rule IV. is amended by the addition of the following sentence:

"The Council may at any time appoint a trustee or trustees, to hold property, real or personal, for the use and benefit of the Institute, upon such terms and conditions as the Council may determine."

III.

To the present Rules, the following is added as Rule IX.

"The Council is hereby authorized to cause to be created under the statutes of the State of New York, a corporation to hold and administer, for the use and benefit of the Institute, such real or personal property as the Council may, from time to time, transfer to it, such transfer being hereby authorized; said corporation not to be subject to control by the members and associates of the Institute, except through the Council, and in such way as may be permitted by the articles of its incorporation, approved by the Council. *Provided*, that the regular business of the Institute, namely, the holding of meetings, and the preparation and distribution of publications, shall remain completely in the control of the Council; and that the funds of the Institute, derived from the fees and dues of members and associates or the sale of publications, shall not be transferred to, or held or controlled by, such corporation, except so far as the Council of the Institute may appropriate money for the necessary expenses of the same—the purpose of this provision being, that the regular business of the Institute, as above defined, shall remain in the hands of the Council, without interference on the part of the said corporation.

"And the Council is hereby authorized to give its assent and co-operation to the generous offer of Andrew Carnegie, a member of this Institute, to provide a building in the city of New York for the use of American Engineering Societies; and in case a corporation should be formed to own and administer such building, the Council is authorized to name representatives for the Institute in the managing board of said corporation."

The Report of the Scrutineers was received, and the result was announced by the President, as follows:

PRESIDENT.

JAMES GAYLEY, ...	New York City.

VICE-PRESIDENTS.

(To serve two years.)

JULIAN KENNEDY, ...	Pittsburg, Pa.
CHARLES D. WALCOTT, ...	Washington, D. C.
GEORGE W. MAYNARD, ...	New York City.

MANAGERS.

(To serve three years.)

F. LOUIS GRAMMER, ...	Baltimore, Md.
CHARLES H. SNOW, ...	New York City.
JOSEPH HARTSHORNE, ...	Pottstown, Pa.

TREASURER.

FRANK LYMAN, ...	New York City.

SECRETARY.

R. W. RAYMOND, ...	New York City.

Mr. James Gayley, the President elect, was introduced by President Ledoux, and, in brief appropriate remarks, accepted the honor and responsibility of the office.

The meeting was then adjourned, with notice that formal supplementary sessions might be held during the pending excursion of many of the members to Cuba, Porto Rico, etc.

MEMBERS AND ASSOCIATES ELECTED.

The following persons have been elected members or associates by postal ballot since the date of the last similar announcement in the proceedings of the New York meeting of October, 1903.

(In response to Circular No. 7, December 17, 1903.)

MEMBERS.

Robert Bell, State Inspector of Mines, . . .	Boise, Idaho.
John M. Brooks, Jr., Mine Surveyor, . . .	Isabella, Tenn.
Elmer Z. Burns, Mine Manager, ...	New York, N. Y.
George W. Bryant, Mine Manager, . . .	Guanajuato, Mexico.
Louis S. Cates, Mining Engineer, . . .	Newton Centre, Mass.
Fred L. Clemens, Draftsman, ...	Sonora, Mexico.
Harold McCloud Cobb, Mining Engineer, . .	Morristown, N. J.

Frank L. Cole, Mining Engineer,	Vladivostok, Siberia.
Edgar A. Collins, Mining Engineer,	Denver, Colo.
Frederick H. Dakin, Jr., Mining Engineer,	Berkeley, Cal.
Walter M. Drury, Mining Engineer,	Butte, Montana.
William Jackson Elgin, Civil and Mining Engineer,	Kanawha Falls, W. Va.
John H. Frazer, Mining Engineer,	Concepcion del Oro, Mexico.
Edward F. Gray, Superintendent Mining Co.,	Ely, Nevada.
Silas A. Knowles, Mine Superintendent,	Idaho Springs, Colo.
Andrew G. Larson, Superintendent Mining Co.,	Rossland, Canada.
Robert Livermore, Mining Engineer,	Ouray, Colo.
Frank W. Maclennan, Mining Engineer,	Cornucopia, Oregon.
Harold M. McLaughlin, Mining Engineer,	Lead City, South Dakota.
Alfred J. Moses, Professor of Mineralogy,	New York, N. Y.
Ben. N. Norton, Mining Engineer,	Cananea, Sonora, Mexico.
Charles S. Palmer, Chemist,	Anaconda, Mont.
George W. Paymal, Mining and Metallurgist,	San Francisco, Cal.
Carlton H. Plumb, Mining Engineer,	Denver, Colo.
Haven Sawyer, Mining Engineer,	Gazelle, Cal.
Jesse C. Scobey, Mining Engineer,	Denver, Colo.
Harry Schoonmaker, Chem., Met. and Assayer,	Cananea, Mexico.
George A. Schroter, Mining Engineer,	Denver, Colo.
Thomas Henry Selby, Mining Engineer,	Mexico City, Mexico.
William W. Smalley, Mine Superintendent,	Chihuahua, Mexico.
Albert M. Smoot, Assayer and Chemist,	New York, N. Y.
Timothy W. Sprague, Contracting Engineer,	New York, N. Y.
Charles A. Straw, Mechanical Engineer,	Wilkesbarre, Pa.
Gustavus A. Swanquist, Surveyor,	Copala, Mexico.
David R. Thomas, Civil and Mining Engineer,	Rossland, B. C., Canada.
Thomas H. Watkins, Coal Operator,	Scranton, Pa.
J. Howard Wainwright, Chemist,	New York, N. Y.
J. Andrew Wauchope, Mining Engineer,	Descanso, Cal.
Marcus White, Mine Manager,	Silver City, Idaho.
Frederick E. Woodbury, Manager Coke & Gas Co.,	Milwaukee, Wis.
Arthur H. Woolrich, Civil Engineer,	Oaxaca, Mexico.
Louis A. Wright, Mining Engineer,	El Paso, Texas.
Wayland H. Young, Mining Engineer,	Etna Mills, Cal.

Associates.

James C. Agnew,	New York, N. Y.
Charles H. Lane,	Cleveland, Ohio.
William F. R. Mills,	Denver, Colo.
Roger T. Pelton,	Poughkeepsie, N. Y.
Sheldon Smillie,	New York, N. Y.

Change of Status, from Associate to Member.

J. Morton Fitzgerald,	Catasauqua, Pa.
L. Webster Wickes,	Anaconda, Mont.
Geo. C. Winslow, Jr.,	Boston, Mass.

From the date of Circular No. 7, 1903 (December 17th) to

February 16, 1904, more than a hundred applications for membership have been received, which are held until final action concerning them has been taken.

Members and Guests Registered.

The following list, which in all probability does not contain the names of all who attended the sessions, is composed of the names registered at the hotel headquarters.

Dr. Albert,	Berlin, Germany.
W. A. Barrows, Jr.,	Sharpsville, Pa.
F. E. Bachman,	Port Henry, N. Y.
Mrs. F. E. Bachman,	" "
Cyrus Borgner,	Philadelphia, Pa.
Edgar S. Cook,	Pottstown, Pa.
Mrs. Edgar S. Cook,	" "
R. H. Chapman,	Washington, D. C.
David T. Day,	" "
Theodore Dwight,	New York, N. Y.
B. F. Fackenthal, Jr.,	Easton, Pa.
Mrs. B. F. Fackenthal, Jr.,	" "
Stanley G. Flagg, Jr.,	Philadelphia, Pa.
James Gayley,	New York, N. Y.
Halbert P. Gillette,	" "
Henry D. Hibbard,	" "
Mrs. Henry D. Hibbard,	" "
L. Holbrook,	" "
Mrs. L. Holbrook,	" "
Miss Holbrook,	" "
Mrs. Justis,	" "
J. E. Johnson,	Longdale, Va.
Mrs. J. E. Johnson,	" "
Charles Kirchhoff,	New York, N. Y.
H. M. Lane,	Cleveland, Ohio.
B. B. Lawrence,	New York, N. Y.
Mrs. B. B. Lawrence,	" "
Albert R. Ledoux,	" "
J. Harry Lee,	Baltimore, Md.
Charles A. Liddell,	Denver, Colo.
Jawood Lukens,	Philadelphia, Pa.
Mrs. Jawood Lukens,	" "
Charles F. McKenna,	New York, N. Y.
Charles A. Matcham,	Allentown, Pa.
E. C. Means,	Low Moor, Va.
E. H. Messiter,	New York, N. Y.
Richard Moldenke,	" "
Mrs. Richard Moldenke,	" "
J. King McLanahan,	Hollidaysburg, Pa.
J. King McLanahan, Jr.,	" "
Robert Nye,	Placerville, Idaho.

George Ormrod,	Emaus, Pa.
M. Otagawa,	New York, N. Y.
Alexander E. Outerbridge, Jr.,	Philadelphia, Pa.
E. W. Parker,	Washington, D. C.
Richard Peters, Jr.,	Chester, Pa.
S. M. Pitman,	Providence, R. I.
Mrs. G. H. Potts,	Pottstown, Pa.
R. W. Raymond,	New York, N. Y.
Mrs. R. W. Raymond,	" "
T. A. Rickard,	" "
James P. Roe,	Pottstown, Pa.
Mrs. James P. Roe,	" "
J. M. Sherrerd,	High Bridge, N. J.
Oberlin Smith,	Bridgeton, N. J.
Harrison Souder,	Philadelphia, Pa.
Henry Souther,	New Haven, Conn.
Mrs. Henry Souther,	" "
H. H. Stoek,	Scranton, Pa.
Joseph Struthers,	New York, N. Y.
Knox Taylor,	High Bridge, N. J.
Mrs. Knox Taylor,	" "
L. H. Taylor,	Philadelphia, Pa.
C. Weichett,	Leipsig, Germany.
H. Vincent Wallace,	Altar, Mexico.
Walter Harvey Weed,	Washington, D. C.
Olof Wenstrom,	Boston, Mass.
Walter Wood,	Philadelphia, Pa.
Frank M. Zeller,	" "

Proceedings of the Eighty-Seventh Meeting, Lake Superior, September, 1904.*

The first session was held at the Assembly Room of the Commercial Club, Duluth, Minn., Wednesday, September 14, 1904, at 9.30 a.m.

President James Gayley called the meeting to order. A speech of welcome was made by the Mayor of Duluth, Dr. M. B. Cullom, followed by an address by Hon. Joseph B. Cotton, both felicitating the Institute on its visit to the Lake Superior region, and expressing the hearty pleasure and compliment that the people of Duluth felt in receiving the members. Mr. Gayley and Dr. Raymond responded in behalf of the Institute.

The Secretary announced the news, received by telegraph, of the death on September 12, 1904, of Mr. Philip W. Moen, a Vice-President of the Institute; and during the subsequent excursions news was received of the death on September 17, 1904, of Mr. Oliver Williams, of Catasauqua, Pa., who for many years was an active and beloved member of the Institute.

The following papers were presented by their authors:

Biographical Notice of William Henry Pettee, by R. W. Raymond, New York, N. Y.

Centrifugal Ventilators, by R. V. Norris, Wilkes-Barre, Pa.

The following papers were presented in oral abstract by the Secretary in the absence of their authors:

Biographical Notice of Sir Clement Le Neve Foster, by T. A. Rickard, New York, N. Y.

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* The exigency of limited space in the present volume of the *Transactions* has made it necessary to condense this report as much as possible. The complete report, numbering 56 pages, was printed and distributed in the Bi-Monthly Bulletin No. 1, January 1905; it contains, in addition to the material here given, the lists of the Local Committees at Duluth, Hougton, Ishpeming and St. Louis; an illustrated description of the steamer-trip on the Great Lakes from Buffalo to Duluth; the train-trip to the iron-ore ranges of Minnesota and Michigan; the excursions at Duluth and environs; the visit to the mines and the furnaces in the copper-region at and near Houghton; the inspection of the iron-mines at Ishpeming and the iron blast-furnace and chemical plants at Marquette; and the concluding excursion to the Exposition at St. Louis.

The Case of Henry Cort, by Charles H. Morgan, Worcester, Mass.

The Genesis of the Diamond, by Gardner F. Williams, London, England.

The Secretary gave notice that, at the next annual meeting, an amendment to the Rules would be proposed, substituting for Rule IX. the following:

"The Council is hereby authorized to incorporate the Institute under the laws of the State of New York, and the Trustees of the corporation thus formed are authorized to adopt rules for the management of the Institute, and to execute such agreements as may be in their judgment necessary and advisable to enable the Institute to participate in the results of the generous offer of its member, Mr. Andrew Carnegie, to provide a building in the city of New York for the use of American engineering societies."

The second session was held at the same place, Thursday, September 15, 1904, at 9.30 a.m., and was devoted largely to papers on blast-furnace practice.

The following paper was read, in the absence of the author, by Mr. F. S. Witherbee, Port Henry, N. Y.

Special Forms of Blast-Furnace Charging-Apparatus, by T. F. Witherbee, Durango, Mexico.

The following paper was presented in oral abstract by the Secretary, in the absence of the author:

Improvements in the Mechanical Charging of the Modern Blast-Furnace, by David Baker, Philadelphia, Pa.

These two papers were discussed together, the following gentlemen taking active part: Messrs. James Gayley, R. W. Raymond, B. F. Fackenthal, Jr., Julian Kennedy, D. T. Croxton, and Prof. B. Osann of Zollenfeld, Hartz.

The following paper was then read by the author:

A Summary of Lake Superior Geology with Special Reference to Recent Studies of the Iron-Bearing Series, by C. K. Leith, Madison, Wis.*

A short discussion followed the reading of the above paper, in which Messrs. Thomas F. Cole, Nelson P. Hulst and W. N. Merriam participated.

The Secretary read by title the following papers, and pre-

* Manuscript not received in time for publication in this volume.

sented them in printed form for distribution among those attending the session;

Appraisal of the Value of Mineral-Lands with Especial Reference to Coal-Lands, by H. M. Chance, Philadelphia, Pa.

The Concentration of Gold and Silver in Iron-Bottoms, by Myrick N. Bolles, New York, N. Y.

The Commercial Wet Lead-Assay, by H. A. Guess, Silverton, Colo.

The Investigation of Alaska's Mineral Wealth, by Alfred H. Brooks, Washington, D. C.

Superficial Blackening and Discoloration of Rocks Especially in Desert Regions, by William P. Blake, Tucson, Ariz.

Report of a Committee to Co-Operate in Standardizing Abbreviations, Symbols, Punctuation, Etc., in Technical Papers.

The Equipment of a Laboratory for a Smelting-Plant, by Herbert Haas, Ingot, Cal.

Evidences of Plication in the Rocks of Cananea, Sonora, by William P. Blake, Tucson, Ariz.

The Manufacture of Coke in Peru, by J. Morgan Clements, New York, N. Y.

Cyanide Practice at the Maitland Properties, South Dakota, by John Gross, Maitland, So. Dak.

Notes on the Flow of Gas from Orifices, by W. R. Crane, Lawrence, Kan.

Crushing in Cyanide Solution as Practiced in the Black Hills, South Dakota, by Charles H. Fulton, Rapid City, So. Dak.

A Geological Cross-Section of the Western Cordillera along the Rio Huasco, by Sydney H. Loram, Chile, S. A.

The Gold Mines of the San Pedro District, Cerro de San Pedro, State of San Luis Potosi, Mexico, by George A. Laird, Sonora, Mexico.

The Taviche Mining District near Ocotlan, State of Oaxaca, Mexico, by H. M. Chance, Philadelphia, Pa.

The Geology of the Treadwell Ore-Deposits, Douglas Island, Alaska, by Arthur C. Spencer, Washington, D. C.

Discussion of Mr. Edwards' paper, Concrete in Mining and Metallurgical Engineering, by F. T. Havard.

Discussion of Mr. Bolles' paper, Concentration of Gold and Silver in Iron-Bottoms, by Edward Keller, Baltimore, Md.

Discussion of Prof. Blake's paper, Superficial Blackening and Discoloration of Rocks, Especially in Desert Regions, by Prof. Theodore B. Comstock, Los Angeles, Cal.

The following papers and contributions, not named above, were read by title at the sessions, for subsequent publication and distribution:

The Occurence of Pebbles, Concretions and Conglomerate in Metalliferous Veins, by Edward Halse, Columbia, S. A.*

The Constitution of Mattes Produced in Copper-Smelting, by Allan Gibb and R. C. Philp, Queensland, Australia.*

The Genesis of the Copper-Deposits of Clifton-Morenci, Arizona, by Waldemar Lindgren, Washington, D. C.

The Decomposition and Formation of Zinc Sulphate by Heating and Roasting, by Prof. H. O. Hofman, Boston, Mass.

Refractoriness of Some American Fire-Brick, by R. F. Weber, St. Louis, Mo.

Labor-Saving Appliances in the Works Laboratory, by Edward Keller, Baltimore, Md.*

Roasting and Magnetic Separation of a Blende-Marcasite Concentrate, by Prof. H. O. Hofman, Boston, Mass.

The Effect of Silver on the Chlorination and Bromination of Gold, by Prof. H. O. Hofman, Boston, Mass.

An Automatic Stock-Line Recorder for Iron Blast-Furnaces, by J. E. Johnson, Jr., Longdale, Va.†

Notes on the Physical Action of the Blast-Furnace, by J. E. Johnson, Jr., Longdale, Va.†

Discussion of Mr. Witherbee's paper, Special Forms of Blast-Furnace Charging-Apparatus, by B. F. Fackenthal, Jr.†

The third (supplementary and final) session was held in the Mines and Metallurgy Building, Block 74, St. Louis Exposition, Mo., on Saturday, September 24, 1904, at 2 p.m.

The meeting was called to order by Mr. Arthur Thatcher, Chairman of the Local Committee, who requested ex-President Robert H. Richards to preside. Mr. Richards called upon Prof. Joseph A. Holmes, who extended, in the name of the Governing Board of the Exposition, a hearty welcome to

* Revised proof not received in time for publication in this volume.

† Manuscript not received in time for publication in this volume.

the members and friends of the Institute, to which Dr. R. W. Raymond responded in behalf of the Institute.

The following papers were presented in oral abstract by their authors:

The Zinc-Smelting Industry of the Middle West, by Herman C. Meister, St. Louis, Mo.

Ore-Dressing Practice in Missouri, by O. M. Bilharz, Flat River, Mo.*

The Fire-Clays of Missouri, by H. A. Wheeler, St. Louis, Mo.

The following papers were read by title by the Secretary:

The Coal-Fields of Missouri, by B. F. Bush, St. Louis, Mo.

Blast-Furnace Plant of the "Elba" Società Anonima di Miniere è di Alti Fòrni at Portoferraio, Elba, by Carlo Massa.

The Relation of Mining Engineering to Other Fields, by Robert H. Richards, Boston, Mass.†

Problems of American Mining Schools, by Samuel B. Christy, Berkeley, Cal.†

Mining Engineering and Mining Law, by James D. Hague, New York, N. Y.†

The following papers, first presented at the New York meeting of the Iron and Steel Institute, October, 1904, were read by title for future publication, under a mutual agreement between the Councils of the two Institutes:

Acid Open-Hearth Manipulation, by Andrew McWilliam and William H. Hatfield, Sheffield, Eng.§

Comparison of Methods for the Determination of Carbon and Phosphorus in Steel, by Baron H. Jüptner von Jonstorff (Austria), Andrew A. Blair (United States), Gunnar Dillner (Sweden), and J. E. Stead (England).§

The Influence of Carbon, Phosphorus, Manganese and Sulphur on the Tensile Strength of Open-Hearth Steel, by H. H. Campbell, Steelton, Pa.

The Application of Dry-Air Blast to the Manufacture of Iron, by James Gayley, New York, N. Y.

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* Manuscript not received in time for publication in this volume.

† Read at the Mining Engineering Section of the International Congress of Arts and Science, September 24, 1904, and the title here entered by special permission of the Congress. The manuscripts, however, were not received in time for publication in this volume.

§ Published in the *Bi-Monthly Bulletin*, No. 2, March, 1905, and omitted from this volume.

The Development and Use of High-Speed Tool Steel, by J. M. Gledhill, Manchester, Eng.*

Members and Associates Elected.

The following persons have been elected members or associates by postal ballot since the date of the last similar announcement in the Proceedings of the Atlantic City meeting of February, 1904:

(In response to Circular No. 1, January 29, 1904.)

Members.

Frank Vans Agnew, Assayer, ... London, England.
John J. Blow, Mining Engineer, ... Brooklyn, N. Y.
Moses Blumenkranz, Mining Engineer, ... Metcalf, Ariz.
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W. H. Case, Mine Manager, ... Tlalpujahua, Mexico.
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Burroughs Edsall, Mining Engineer, ... El Paso, Texas.
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William Longworth, Manager of Mining Co., . . Lithgow, New South Wales.
Jasper A. McCaskell, Mining Engineer, ... New York, N. Y.
George W. McNaughton, Manager of Mica Mines, Sydenham, Ont., Canada.
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Harlan S. Miner, Chemist, ... Gloucester City, N. J.
Henry C. Mueller, Miner, ... Morenci, Ariz.
Edward E. Nelson, Mining, ... Guayaquil, So. America.
Masayuki Otagawa, Civil and Mining Engineer, . Tokio, Japan.

* Published in the *Bi-Monthly Bulletin*, No. 2, March, 1905, and omitted from this volume.

Juan de la Cruz Posada, Mining Engineer, Medelin, Colombia, So. America.
Richard Read, Mine Owner, ... Sydney, New South Wales.
Carl von Reischach, Mining and Metallurgical Engineer, San Francisco, Cal.
Paul Reisinger, Mining Engineer, ... Stockett, Mont.
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Chojiro Yokogama, ... Cambridge, Mass.

Change of Status from Associate to Member.

William H. Landers, A. Campbell McCallum, William G. Mather and Frank J. Nagel.

(In response to Circular No. 2, March 1, 1904.)

Members.

Clarence Eugene Abbott, Mining Engineer, . . . Eveleth, Minn.
H. Albert, Mining Engineer, ... New York, N. Y.
Percy P. Barbour, Mining Engineer and Surveyor, Idaho Springs, Colo.
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Ballarat School of Mines, South Australia.
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William Young, Mine Operator, ... Rat Portage, Ont., Canada.

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Change of Status from Associate to Member.

Harry L. Williams.

(In response to Circular No. 3, April 18, 1904.)

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Axel E. Anderson, Draughtsman, Surveyor, . . . Denver, Colo.
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Oscar M. Weichsel, ... New York, N. Y.

Change of Status from Associate to Member.

E. Davenport Cleland, R. Prewitt Coleman, Roger T. Pelton.

(In response to Circular No. 4, June 10, 1904.)

MEMBERS.

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Henry C. Beeler, Mining Engineer, ... Cheyenne, Wyoming.
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Samuel Whitworth, Engineer, ... Kalgoorlie, Western Australia.
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ASSOCIATES.

Marc Bailey, Student, ... Houghton, Mich.
Frank A. Dalburg, Student, ... State College, Pa.
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Ray L. Herrick, Student, ... Lehigh University, Pa.
Hansaburo Hunter, Student, ... London, England.

CHANGE OF STATUS FROM ASSOCIATE TO MEMBER.

Sidney M. Bamberger, Harry J. Wolf.

(In response to Circular No. 5, August 16, 1904.)

MEMBERS.

Rafael Flores Alatorre, Mine Manager, ... Hidalgo, Mexico.
Charles I. Auer, Assayer and Chemist, ... Durango, Mexico.
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Charles Lazarus Bryden, Assistant in Chemistry, . Easton, Pa.
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Frederick D. Dimmick, President of Mining Company, Birmingham, Ala.
Alexander George Dixon, Mining Engineer, . . . San Luis Potosi, Mexico.
William James Dooley, Mine Owner and Operator, Salt Lake City, Utah.
Frederick D. Fuller, Metallurgist, ... Sumpter, Ore.
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John A. Redfern, Mine Inspector, ... Hibbing, Minn.
Arthur F. Rising, Mining Engineer, ... Madoc, Ont., Canada.
John Burgoyne Stewart, Mining Engineer, ... El Oro, Mexico.
A. A. Talmage, Civil Engineer, ... Los Angeles, Cal.

CHANGE OF STATUS FROM ASSOCIATE TO MEMBER.

James Carson Agnew, Sidney M. Bamberger, W. Arthur Sloan, George O. Smith, George K. Taggert, Loyal W. Trumbull, H. T. Willis, Harry J. Wolf.

PAPERS.

Wet Methods of Extracting Copper at Rio Tinto, Spain.

BY CHARLES H. JONES, SANTIAGO DE CUBA.

(Atlantic City Meeting, February, 1904.)

THE ore treated at Rio Tinto is a massive iron pyrites containing up to 3 per cent. of copper, which has been disseminated through the mass by a secondary enrichment.

The well-known method adopted for the extraction of the copper consists simply in allowing huge heaps of the mineral to oxidize under the influences of moisture and air, and subsequently washing out the copper sulphate as soon as it is formed, by running water through the heap.

At the outset it may be well to have it clearly understood that the successful and economical application of this system depends largely on the state in which the copper occurs in the mineral. If it exists as chalcopyrite, $CuFeS_2$, the copper will not oxidize by simple exposure to the air, in one case it having taken many years to oxidize 10 per cent. of the copper originally present in the ore. If the copper is in the form of CuS, the oxidation proceeds very slowly. The best form for solution is Cu_2S, or copper glance, which constitutes the bulk of the copper in Rio Tinto pyrite. These statements may be made clearer from the following account of the reactions that take place during the oxidation.

When the mineral is exposed to free access of air and moisture, some ferrous sulphate is formed in accordance with the following reaction:

$$(1) \qquad FeS_2 + 7O + H_2O = FeSO_4 + H_2SO_4.$$

This ferrous sulphate becomes readily oxidized by the air to ferric sulphate.

$$(2) \quad 2FeSO_4 + H_2SO_4 + O = Fe_2(SO_4)_3 + H_2O,$$

and it is due to the action of this ferric sulphate on the copper

sulphides that the copper is rendered soluble, as is shown by the following chemical equations:

$$(3) \quad Fe_2(SO_4)_3 + Cu_2S = CuSO_4 + 2FeSO_4 + CuS.$$

$$(4) \quad Fe_2(SO_4)_3 + CuS + 3O + H_2O = CuSO_4 + 2FeSO_4 + H_2SO_4.$$

The reaction No. 3 takes place fairly rapidly and causes half the copper to go into solution within a few months, while the reaction No. 4 proceeds much more slowly, and requires, under the most favorable conditions, about two years to extract 80 per cent. of the remaining half of the copper.

Here I may add that in the laboratory at Rio Tinto (under the direction of Mr. W. A. Jenkin), a method has been worked out to determine the state of combination of the copper in any particular mineral. This method depends on the action of the mineral on various solutions under constant conditions of dilution and temperature, and, though necessarily somewhat arbitrary, it shows with considerable accuracy the form of sulphide in which the copper exists, and consequently whether the copper can be readily extracted by washing in heaps.

In practice the method adopted to bring about the desired oxidation is as follows: A site is chosen for the formation of the heap where the ground is sufficiently concave and sloping to enable the copper liquor that is formed to collect and run out at the base of the heap. On the ground is first arranged a network of air-flues, made of rough stones and having an internal diameter of 12 inches. Vertical chimneys, 50 ft. distant from one another, are built in the same manner and connect with the ground flues. Care is taken that the mouths of the ground flues are kept open and not covered by the ore. The mineral, the lump portion of which has been passed through jaw-breakers to be reduced to pieces not larger than from 2 to 3 in. across, is now tipped from side-tip wagons at the highest part of the selected site over and around the stone flues. "Lump" and "fines" are alternately dumped until the height of the mass at the edge is about 30 ft., the upper surface of the mineral being kept level. A heap of this form approximately contains 100,000 tons of ore.

As the mineral is added, the building of the stone chimneys

keeps pace, in order to have a clear opening to the top of the heap. The surface of the heap is formed into squares by means of ridges of the mineral, the size of these squares depending on the porosity of the heap. The function of these ridges is to enable the water to be run on locally over the surface of the heap in order to insure that all parts are equally washed and that the water does not run through the heap in channels. A system of gutters is also arranged so that the water can be run on to all parts of the mass. As the heap is being formed, water is run on, and the copper sulphate existing in the mineral is extracted; the water also provides the moisture necessary for the oxidation in accordance with the equations given above. The mineral in the heap is then allowed to oxidize, which it does pretty rapidly, as evidenced by the heat produced, the temperature in the chimneys rising to from 170° to 180° F. As the temperature increases, the surface-openings of the chimneys may be closed in order to allow the oxidation to spread through the heap. The surface gradually shows a brownish coloration due to the dehydration of the buff-colored basic ferric salt that forms on the top of the mass, and its gradual heating-up may be noted by this drying action. The greatest care must be taken not to allow the heap to fire, for a fire once started is very difficult to extinguish. When the oxidation has proceeded as far as it is safe to allow it, water is run on at the rate of about 50 cu.m. per hour, until the soluble copper is leached out; the heap is then allowed to re-oxidize and the washing is repeated. After about a year has elapsed the surface requires "retilling," and the squares are re-arranged so that the places where the ridges were before are now the middles of the squares; the gutters also are shifted. At the edge of the heap for a distance of some yards, the mineral, which has become cemented, holds a considerable quantity of copper salts and is dug down into terraces in order that this copper may be extracted by washing. When the copper is reduced to 0.3 per cent., the heap is considered washed and the mineral, containing 49.5 per cent. of sulphur, is removed and exported as "washed sulphur-ore," and utilized for the manufacture of sulphuric acid. Fig. 1 shows the surface of the heaps arranged for washing.

Successful heap-washing depends on the efficient ventilation

of the mass, the trouble usually being a too great excess of "fines" produced in mining the ore, which cement hard and clog up the air passages.

The copper liquor as it runs from the heap contains some ferric iron in solution, which, as will be shown later, is very objectionable. In order to remove the ferric iron the liquor is run over a smaller heap of fresh mineral known as a "filter bed" which reduces the ferric iron. This "bed" is laid inside a reservoir formed by a masonry dam across a small ravine, and the liquor after percolating through the mineral remains in contact with it until it is required to be drawn off to the precipitating tanks. When the mineral is fresh, the reduction of the ferric iron takes place rapidly, due to the Cu_2S as shown by equation 3, but the iron pyrites itself has an effective reducing action on ferric iron in solution according to the following equation:

$$(5)\quad 7Fe_2(SO_4)_3 + FeS_2 + 8H_2O = 15FeSO_4 + 8H_2SO_4.$$

A general view of the precipitating tanks and the masonry dam forming the reservoir for the percolation of the copper liquor is shown in Fig. 2. The principal constituents of the liquor as it enters the cementation tanks are as follows, the figures given representing the grams per cu. m. or units per million parts:—Copper, 4,000; ferric iron, 1,000; ferrous iron, 20,000; free sulphuric acid, 10,000; and arsenic, 300. The large quantities of ferrous iron and free sulphuric acid present are due to the fact that the waste liquor from the cementation tanks after the copper has been precipitated ("salida" liquor as it is called), is pumped back and used for washing the heaps in addition to fresh water, and consequently these solutions tend to become concentrated. The liquor is then run from the reservoirs at about 300 cu.m. per hour through the precipitation tanks over pig-iron in order to precipitate the copper in the form of so-called "cement copper" or "copper precipitate." These cementation tanks are arranged in series on the slope of a hill, the liquor passing backwards and forwards until it is discharged from the lowest tank of the series free from copper.

Each series consists of three tanks in parallel arranged so that the liquor can be divided up and passed along as many

Fig. 1.

Surface of Heaps Arranged for Washing.

Fig. 2.

General View of Precipitating Tanks, Copper Liquor Dam and Min

FIG. 3.

VIEW SHOWING THE METHOD OF REMOVING THE CEMENT-COPPER FROM THE PRECIPITATING TANKS.

tanks as necessary, depending on the quantity of liquor that is being run through and on the varying temperature of the liquor with different seasons, the hotter the solution, which in summer reaches 100° F., the faster the rate of precipitation. Each tank is about 320 ft. long, 5.5 ft. wide and 2.25 ft. deep and has a slope varying from 2 per 1,000 in the first series to 11 per 1,000 in the last, the reason for the increase in slope being, that as the liquor becomes impoverished in copper, the free acid present is more active in wastefully dissolving the pig-iron,—an action which is considerably diminished by increasing the velocity of the liquor by means of the increased slope of the tanks. The tanks themselves are made of 9- by 3-in. boards attached to wooden frames set in cement, the space between parallel tanks being filled in with stone and cement, constituting a wall supporting the sides of the tanks. A view of the tanks showing the method of removing the cement copper is shown in Fig. 3.

No metal is used in the construction, hard wooden pegs being employed to attach the boards to the frames. The spaces between the boards are carefully calked with oakum and pitch in order to render the tank water tight. At each end of the tank is an arrangement by which a door can be dropped in and luted so as to cut out that particular tank, and there are also wooden plugs that can be removed so that the liquor from that tank can be run off, thus allowing for the removal of the precipitated copper. A few old boards are placed on the bottom of the tanks for their protection and on these are piled up the pigs of iron which are laid across the tank at the bottom, the next layer being at right angles to the first, and so on until the tank is filled; each foot-length of the tank contains about one ton of iron. The liquor is allowed to run through the system of tanks and needs no attention except to remove the precipitated copper and to add fresh iron. The "salida" liquor containing from 15 to 20 grams of copper per cu.m. is allowed to run to waste, for the reason that with this copper-content the amount of iron required to precipitate the copper equals the value of the copper recovered. Daily some of the tanks are cleaned out by being closed as described above, the liquor meanwhile passing down the other tanks of the series; the liquor is run off into settling tanks, any copper in suspension being there recovered; all of the iron is removed from the tank and piled on to the dividing wall, at the same time the copper adhering to the iron is knocked off and thrown back into the tank. The dirty looking precipitate is then transferred to the cleaning and concentrating plant, the iron is replaced in the tank and the liquor again allowed to run through it. This crude precipitate, containing about 70 per cent. of copper, is thrown, a little at a time, on to a perforated copper plate placed at the head of a long launder or tank and is washed through the plate by a strong stream of water from a small nozzle. The material that does not pass through the screen consists of leaf-copper and small pieces of iron; this material is thrown into a heap and afterwards sorted over by girls who remove the pieces of iron. The precipitate that passes into the launder is repeatedly turned over against the stream of water and by this simple means a concentration is effected.

The first few yards of the launder contain a red precipitate known as "No. 1 precipitate," containing 94 per cent. of copper and less than 0.3 per cent. of arsenic; following this is "No. 2 precipitate," containing 92 per cent. of copper and between 0.3 and 0.75 per cent. of arsenic, while below is the "No. 3 precipitate;" this is in a state of very fine division and contains on an average 50 per cent. of copper and 5 per cent. of arsenic. This last-named portion, which carries all the graphite from the pig, contains the bulk of the antimony and bismuth that is also precipitated from the liquors. Classes Nos. 1 and 2 are removed to the drying sheds and are bagged for shipment to the refinery; the No. 3 precipitate is removed, moistened with acid liquors, made into balls by hand and dried in the sun. These balls become cemented hard and can be readily transferred to the smelter, where they form part of the charge for the blast-furnaces and are run down to matte to be subsequently bessemerized, thus effectively removing the arsenic, antimony and bismuth that they contain.

The reactions that take place in the cementation tanks are given in equations 6, 7 and 8.

The first reaction that occurs in the liquor running over metallic iron is the reduction of the ferric sulphate to ferrous sulphate, the final result being in accordance with the equation:—

$$(6) \qquad Fe_2(SO_4)_3 + Fe = 3FeSO_4.$$

This action causes the consumption of the pig-iron without any corresponding yield in copper, and consequently should be avoided as far as possible by having all the iron in the ferrous state.

The second reaction is the precipitation of the metallic copper, brought about by galvanic action. The iron becomes coated with copper, and thus the iron and copper in the acid liquor constitute a galvanic couple with a considerable difference of potential. It is due to the electrolytic action that the copper and all other metals present that are electronegative to iron will be precipitated. The ultimate action of the precipitation may be chemically expressed by the following equation:—

$$(7) \qquad Fe + CuSO_4 = FeSO_4 + Cu.$$

Besides the reactions above mentioned there is one which causes the liberation of hydrogen, as evidenced by the bubbles of gas that may be observed to arise in the tank liquor. This action, which causes a wasteful consumption of the iron, may be expressed as a final result by the following equation:—

$$(8) \qquad Fe + H_2SO_4 = FeSO_4 + H_2.$$

These three equations constitute the main reactions that take place in the precipitating tanks.

While the liquor is fairly strong in copper, the copper is mostly precipitated in a coherent form, but in the later stages as the liquor becomes impoverished it is precipitated in a powdery state—a condition which is more effective in its galvanic action with the iron, and thus unfortunately causes a larger precipitation of arsenic and other impurities than in the earlier stages. In the later stages also the "solution reaction" of iron in sulphuric acid as given in reaction No. 8 goes on to a proportionately greater extent than does the precipitation of copper, and consequently the cost of pig-iron in precipitating the copper varies inversely as the quantity of copper in the liquor.

By keeping careful watch on the reduction as far as possible of the ferric iron before the liquor enters the tanks, and by giving it sufficient velocity through the tanks, a strongly acid liquor such as given above during a year's working will not consume more than 1.4 units of pig-iron (containing 92 per cent. of iron) to 1 unit of copper precipitated. A valuable check on the iron being consumed can easily be kept by the laboratory, by analyzing the liquors before entering and after leaving the tanks, and from these analyses the quantity of iron that is being consumed can be calculated.

This description of the methods of leaching and precipitation that are now used at Rio Tinto is offered in the hope that it will prove of interest and value. Especially so in view of the fact that most of the information heretofore published pertaining to these methods has become obsolete.

Notes Upon Preliminary Tests and Cyanide-Treatment of Silver-Ores in Mexico by the MacArthur-Forrest Process.

BY JOHN F. ALLAN, CITY OF MEXICO.

(Atlantic City Meeting, February, 1904.)

THIS paper does not pretend to advance any facts or improvements not known to many members of the Institute, but is intended merely to give a few practical hints on preliminary tests, and to call attention to a possible enlarged field for the cyanide process in the treatment of some silver-ores, which merits investigation. Broadly considered, the choice of a process for the treatment of silver-ores in Mexico is confined to the following alternatives: (1) Concentration for treatment by smelting; (2) milling and pan-amalgamation; (3) the *patio* and Ortega process; (4) chloridizing roast and lixiviation; (5) chloridizing roast and cyaniding; and (6) cyaniding of the crude ore.

These methods of treatment may also be combined, and cyaniding or concentration, or both, may be introduced, at whatever stage it is found to be most advantageous.

The characteristics of the ore (especially its gold-content) and local conditions, such as distance from railway, cost of fuel, amount of water available, vary so much that it is impossible to make general comparisons between the different processes, either as to extraction or cost. Concentration and subsequent smelting may be made very expensive by long-distance transportation of concentrates; pan-amalgamation is undoubtedly expensive; and the *patio* process is barred when the ore contains appreciable amounts of gold, by reason of its poor extraction of that metal; yet there are many ores for which each of these processes is especially adapted, and the expense of treatment is compensated by the results obtained.

On the other hand, the residues of concentration, amalgamation, *patio* and lixiviation have been found in many cases amenable to cyaniding; and although this does not prove that cyaniding could have been advantageously used otherwise than as a supplementary process, yet it certainly makes that question worth investigating.

Silver is generally found in those ores either native or as a chloride or possibly chloro-bromide, associated with carbonates, in surface and oxidized portions of veins. In depth, and removed from the oxidized zone, the metal usually appears combined with sulphur, antimony and arsenic, as well as associated with galena, blende, pyrites and other minerals.

Experience in testing a large number of silver-ores proves that the antimonides, fahlerz, and more especially ores in which the silver is isomorphously associated with galena and other sulphide ores, are not amenable to the cyanide treatment as at present known. Ores of the latter class are by far the most abundant; and since, even in the most favorable cases, the silver is associated with minerals injurious to cyaniding, it is not surprising that the scope of the process has been very limited, by reason of low extraction and heavy cyanide-consumption. On the other hand, some oxidized ores, and some ores containing sulphides of silver (probably the purer varieties), have given good extraction even with comparatively weak solutions, and at a consumption of cyanide which, though heavy as compared with that of most gold-ores, lies within the limits of economical treatment. Moreover, and to a much greater extent, it has been found that, when the "cyanicides" are rendered innocuous and the silver soluble by means of a roast, especially a chloridizing roast, cyaniding can be adopted with great advantage, particularly when the ore contains, as it very often does, an appreciable amount of gold. The greater the proportion of gold, the more manifest the advantage of cyaniding; hence the advisability of making careful preliminary tests before concluding that this process is unsuitable. Whether an ore is amenable to cyaniding cannot be decided by its physical characteristics alone. The purest and most harmless-looking quartz-ores are sometimes the most refractory, while ores, which the presence of base metals renders unpromising, may yield satisfactory results.

As silver occurs combined with other substances, its treatment by cyanide presents much greater difficulty than that of gold, and hence the necessity of the most complete preliminary tests both in the laboratory and on a working scale before definitely adopting that process. Although a chloridizing roast is sometimes prima facie a necessity, yet when it is considered that the losses of silver by volatilization amount to 5 per cent. in the most favorable case and rise to 10 and 15 per cent., and that the cost of roasting is not below $1 per ton in very favorable cases, and amounts to $3 and even $4 where local circumstances are adverse, it will be easily seen that every endeavor should be made to treat the ore raw, even at some sacrifice of extraction, and that chloridizing should be adopted only as a last resort.

Investigation of Samples and Preliminary Tests.

In examining a sample, the procedure found most advantageous is as follows:

1. Assay, take specific gravity and examine physical and mineralogical characteristics.

2. Test for acidity, latent or active, and cyanide-consumption.

3. Make small agitation-tests, with different strengths of cyanide-solution, upon ore ground so fine as to be retained on 40-mesh and on 80-mesh, and upon the slimes.

These tests establish roughly whether the ore is amenable to cyanide treatment, what will be the cyanide-consumption, to what size of mesh the ore must be ground to give the best extraction, and whether preliminary treatment is necessary. From this investigation tests can be pushed in the direction which holds out the best chances of success; the strength and time of contact of solution can be varied, and the preliminary treatment studied.

Preliminary treatment may be divided into four classes, intended respectively for the correction of acidity, and for the removal of cyanicides by solution, by mechanical means, and by roasting or chloridizing.

The Correction of Acidity.—The test for acidity will have shown the amount of alkaline matter required to make the ore slightly alkaline or neutral. The treatment can vary between

simple water-washes for removing soluble salts, to lime-water or caustic soda, with preliminary and subsequent water-washes in obstinate cases. The easiest and usual manner of neutralizing acidity is by intimately mixing lime with the crushed ore and soaking with sufficient water. In such cases it is advisable to give sufficient time for the reactions before adding cyanide solution.

The Removal of Cyanicides by Solution.—In cases where cyanicides, such as copper carbonate, would cause a heavy consumption of cyanide, preliminary acid-washes have been used. This treatment is made expensive not only by the necessary solutions and the consumption of cyanide after caustic-soda wash, but by the extra handling and plant, since it may be necessary to treat the ore in a different tank from that in which it has been subjected to the acid and subsequent alkaline-wash, in order to prevent the acid from decomposing the solution adhering to the sides and bottom of the tank. If the ore contains lime, the acid-consumption is high, and the charge may pack and prevent leaching. A study of local conditions is necessary, to see whether it would not be preferable to use a dead or chloridizing roast as a remedy.

Removal of Cyanicides by Mechanical Means.—Minerals consuming cyanide, or which will not yield the precious metals associated with them, can perhaps be removed in the first instance by hand-sorting, or, if necessary, by various forms of concentration. If the ore cannot be treated direct, but only after the heavier minerals have been removed by concentration, it becomes a matter of calculation whether the concentrates are valuable enough to be marketable, or, if not marketable, still too valuable to be disregarded; and upon this will hinge the adoption of the treatment.

Removal of Cyanicides by Roasting or Chloridizing.—Ore which cannot be treated raw sometimes gives good results after a dead roast, or more often a chloridizing roast. The former is applicable to cyaniding, when the ore contains gold, and in cases where salt cannot be obtained. When the ore contains no gold and can be chloridized, lixiviation with hyposulphite of soda is often cheaper and more advantageous than cyaniding. In chloridized ores it can be laid down as a general rule, that cyaniding will give as good if not better extraction of silver than lixivia-

tion; but it is only when the gold-content is so great that the increased extraction of that metal counterbalances the extra cost of cyaniding, that this method becomes preferable to lixiviation.

Final Preliminary Tests.

After experiments on a small scale have determined the most suitable method of treatment, final tests can be made to obtain results on a sufficiently large scale to warrant the construction of a plant, and to give sufficient data for its design.

If the preliminary tests prove, as is sometimes the case with silver-sulphide ores, that favorable extractions can only be obtained by grinding to a very fine mesh, the feasibility and cost of reducing the ore to this size should be studied. If it be found impracticable, cyaniding can be confined to the slimes produced in ordinary milling, and a more suitable process can be adopted for sands.

These final tests are necessarily divided into two classes, covering the treatment of sands and slimes respectively.

The Treatment of Sands.—This is carried on in vats from 4 to 5 ft. in diameter and from 4 to 4.5 ft. high, supplied with appropriate filter-bottoms and upper and lower sumps of similar size, and a zinc-box. The tests should establish: (1) the size at which the ore gives the best extraction; (2) the minimum strength of solution necessary; (3) the amount of lime to be added to neutralize acidity or make solutions slightly alkaline, as well as to obtain exact results, if a more elaborate preliminary treatment has been found necessary; (4) the time and bulk of solution necessary; and (5) data for estimating the cost of treatment and the size and nature of the plant required for a given product.

The Treatment of Slimes.—This is carried on in vats 3 or 4 ft. deeper than those used for leaching, with the same complement of sumps and zinc-boxes. Some form of agitation is, however, necessary, such as paddles worked by hand, a small pump, or compressed air, if obtainable. The tests should establish: (1) the rate of settling of the slimes; (2) the amount of lime necessary to settle slimes as rapidly as possible, without making solutions too alkaline; (3) the minimum strength of solution necessary; (4) the periods of activity of solution; (5) the number and bulk of solutions required for the best results; (6) the

number of washes required to remove solutions bearing precious metals; and (7) data for estimating the cost of treatment and the size of plant required. In cases where slimes are rich or settle with difficulty, filter-presses can be used to advantage.

Notes.—In leaching it has been found advisable to distribute the charge evenly in the tanks, so as to prevent packing and subsequent channeling of the solutions. It has been found in practice that adding small lots of solutions at frequent intervals, but allowing the charge to drain between one solution and the next, sometimes gives a better extraction than continuous leaching or soaking and leaching. This is due, no doubt, to the admission of air. In leaching, also, the bottom layer of the tank rarely gives as good an extraction as the upper part. This may be due to insufficient washing, or to the solution being, by reason of fouling, low in dissolved oxygen. It is always advisable to sample and assay the lower and upper parts of the charge, and if they show any appreciable difference, to find out whether the fault is due to insufficient washing, or whether further treatment will increase the extraction. If the bottom layer is found to yield a further extraction, even after thorough washing, the solution can be added by upward percolation, in order to give the charge the necessary amount of oxygen. If this is insufficient, solutions can be aërated by allowing them to drop into the sump in the form of spray, or by agitating them with compressed air; or, finally, the charge can be turned over so that the lower layers shall be on the top. In one special instance, turning over the charge effected an increased extraction of $1 at a cost of 3 c. per ton.

While studying leaching it is necessary to study the solutions and assay them for gold and silver, in order to find out when the extraction takes place, and when it is economical to stop. This is all the more requisite, as in some instances a soak or slow leach is found preferable to rapid leaching.

For both sands and slimes it is absolutely necessary to use the same solution on six or seven consecutive charges, so as to make sure that the solution does not foul. It has been noted in several instances that ores, which to all appearance were harmless and gave good extractions with fresh solution, yielded decreasing extractions with every subsequent charge, the solution gradually losing its solvent power. This fouling of the

solution may be due in the case of old tailings and dumps to organic matter; in the case of ores containing copper, to the gradual increase of double cyanide of copper, with decrease in dissolved oxygen, or to the formation of soluble sulphides with consequent retarding effect upon extraction. In other cases, the cause is obscure. When the fouling is due to organic matter, the remedy in obstinate cases is a roast, if feasible. When copper is the cause, every endeavor should be made to precipitate this metal in the zinc-boxes, so as to prevent its accumulation in the solution. The solutions can also be aërated by mechanical means. Alkaline sulphides can be removed by adding lead carbonate or other lead salts to decompose them; but it is worthy of note that sulphides are rarely formed, probably because of the fact that many silver-ores contain the necessary lead salts to decompose them on formation. The point at which solutions become fouled can also be studied, and fresh solution added, a sufficient amount of that partially fouled being discarded after using it as a preliminary wash and reducing the cyanide-contents as much as possible. Finally, in obstinate cases, a chloridizing roast becomes necessary. The fact of fouling having been established, its remedy must be approached with the utmost caution. Cyaniding must not be definitely adopted until it is unquestionably proved that fouling can be remedied. Repeated tests with the same solution are also to be advocated, as it has been found in some instances that the addition of lime has produced a retarding effect, if not a decrease in silver extractions.

I have frequently tried oxidizing agents, both in the preliminary and in the final tests; but none of these have been found to effect any material improvement in the extraction, and they have, as a rule, considerably increased the consumption of cyanide. This need not, however, preclude further tests in this direction. In some classes of ore improvements may in future be found without any counterbalancing losses in cyanide and increased cost.

In treating slimes, lime is added in the proportion indicated by the preliminary tests, to remedy any acidity, as also to settle the slimes. The excess of water is then decanted, the specific gravity and assay of pulp are taken and the first solution is added. The bulk and strength of the solution are carefully

ascertained, and the portion decanted is tested for cyanide and precious metals. Careful study will thus indicate the chemical consumption of cyanide, the periods of greatest activity in dissolving gold and silver, the number and strength of solutions required, the economical point at which treatment is finished, and the number of weak-solution washes necessary to remove the gold and silver in solution from the pulp. Finally, when the tank has received treatment, it is necessary to assay an accurate sample of the pulp, in order to find the actual extraction, and to wash this sample and assay it again, in order to ascertain whether the charge has had sufficient washing. The assay of the sample of discharge should also approximate the result obtained from the assays on solutions decanted.

In making final tests, whether of sands or slimes, it is most important to check the bullion produced with the amounts called for by assay, as well as to study the precipitation, and obtain the necessary experience for securing good results in the commercial plant. The zinc-boxes should be calculated to hold a sufficiency of zinc for precipitation, and have a capacity for handling a continuous stream of solution. The zinc should be cut in such a manner that the shavings form a homogeneous spongy mass, without being so fine as to lose the toughness of the fiber, or so coarse as to offer insufficient surface. Solutions should be passed through the boxes continuously, and a fair action secured, care being taken not to allow the boxes to stand idle more than is unavoidable, because such idleness permits the formation of zinc salts injurious to precipitation.

Testing the passage of the solution through zinc-boxes is also important for slimes which require a large addition of lime for settling, because, even if the lime does not injure the extraction, it may cause excessive action and foaming in the zinc-boxes, with consequent loss of cyanide and zinc.

In some cases, on the other hand, it has been observed that the precious metals are deposited upon the zinc in metallic form. To prevent this deposit, and the consequent locking up of values, the action in the boxes can be stimulated by increasing the amount of lime added in the tanks.

Solutions are as a rule sufficiently strong to cause no trouble in starting precipitation, and indeed silver precipitation is generally very complete, the sump-solutions showing by assay only

traces. Should it by any chance be found necessary to stimulate precipitation, this can be effected by a strong-solution drip at the head of the box, or by dipping the shavings into a solution of lead acetate.

When the ore contains an appreciable amount of copper and a portion is dissolved by the solution, the problem of precipitation requires great care, in order to prevent accumulation of the double cyanide of copper, and consequent fouling in the solution. Copper is precipitated from strong cyanide solutions in preference to gold, or (to a smaller extent) silver; a drip of strong solution may therefore have the effect of precipitating the copper where required. It is undoubtedly preferable to precipitate the gold and silver first and promote the precipitation of copper in the lower compartments. This can be done by dipping the zinc for copper precipitation in lead acetate, either with or without strengthening the solution. The consumption of zinc in precipitation of silver is not necessarily high; about 1 to 1.25 lb. per ton can be regarded as a usual quantity in ores containing not above 35 oz. silver per ton. The silver product, where not much gold is present, is gray in color, and is refined and melted into bars with facility.

Treatment of Ores Necessitating a Previous Chloridizing Roast.

Roasting should be the subject of exhaustive preliminary experiments, especially for ores containing base metals. Laboratory tests are insufficient. If possible, a small reverberatory should be built for tests on a working scale. In roasting for cyaniding, the object, besides minimizing the loss of precious metals by volatilization, is to obtain a product which will give the highest extraction with the least cyanide-consumption. Such roasting, therefore, requires greater care than the usual chloridizing roast for lixiviation. To secure a good chloridized product for cyaniding, the minerals associated with the ore should be carefully studied, to discover whether any injurious substances are present, and if so, how to remove them or minimize their effects.

In roasting for lixiviation, the higher the chloridization the better the extraction. In cyaniding this is also the case, but it does not follow that for cyaniding the highest chloridization,

involving serious losses by volatilization, would give the best economical results. In one instance, old, refractory pan-amalgamation tailings carrying 1 part of gold to 2 of silver, and showing 30 per cent. extraction when treated raw, gave extractions of over 70 per cent. after roasting in a White-Howell roaster, which chloridized to 70 or 75 per cent. by hypo-test. Precisely similar results were obtained without increase in cyanide-consumption by chloridizing to only 45 or 50 per cent., thus effecting a reduction in working costs and reducing the loss by volatilization. This does not by any means occur frequently, but it is possible, and well worth bearing in mind.

For testing-purposes, ores of this class may be divided into: (1) those in which silver is present in a calcareous gangue, giving an alkaline reaction after chloridizing; and (2) those in which silver is associated with base metal sulphides, giving an acid reaction after chloridizing.

Alkaline Ores.—Ores containing lime and magnesia in considerable quantity do not, as a rule, give favorable results, unless associated with oxide of manganese, or unless it is possible to add pyritic ore as a remedy. This class of ore usually requires a high temperature; and as both lime and magnesia are easily volatilized, they may carry off the precious metals in undue amount. It is also advisable to charge the ore in tanks as soon as possible, to prevent cementing or packing, and to avoid gradual cooling in mass, epecially if the material has been wet, in order to prevent the action of the caustic lime on the chloride of silver.

Ores Containing Base Sulphides.—These should be examined to discover whether it is more advantageous to chloridize the base metals together with the silver, or to give a preliminary oxidizing roast and chloridize the silver subsequently.

In the former case it is immaterial whether salt be added with the charge, or an hour or two later, as the salt is not decomposed before the formation of sulphates commences. The base metals are converted into chlorides, and in volatilizing carry off precious metals, causing loss; but as against this, the ore is partly relieved of their presence, and the remaining portions can be removed by water-wash.

If roasting is carried to the point where the base metals are oxidized before the salt is added, they will hardly be affected

by the chlorine, and a saving may be effected by avoiding the volatilization of the precious metals. Antimony and arsenic, however, when present, are volatilized, causing loss—although the removal of these metals is beneficial. Copper, when present, is formed into an oxide, causing subsequent trouble in cyaniding. In chloridizing ores of this class, gradual cooling in special deposits or heaps, and also wetting down, are sometimes found to increase the degree of chloridization of the precious metals.

Although the previous examination of the ore will have given a general notion of the form of roasting likely to prove most advantageous, yet comparative tests are necessary to place beyond doubt the best method of treatment. These are made (1) by adding salt with the charge and examining the behavior of the ore with different percentages of salt and at various temperatures; and (2) by giving a preliminary dead roast, and then proceeding as above.

In each case the tests should determine the degree of chloridization by hypo-test; the loss in precious metals by volatilization; and the cyanide extraction and consumption for small quantities of each class of material.

Once the most favorable method of roasting is decided upon, cyanide-tests on a working-scale can be made. In this case slimes-treatment becomes unnecessary, as the slimes will have been mixed and chloridized with the sands in the case of dry crushing, or separated, dried, disintegrated and mixed with sands in the case of wet, the roasted ore in every instance affording a suitable product for leaching. Further experiments can be made to ascertain: (1) whether the degree of chloridization is increased by gradual cooling in deposits or heaps, and whether wetting down is injurious or otherwise; and (2) whether a preliminary wash to remove soluble chlorides is advisable.

Cyanide-tests may be carried out in leaching-tanks similar to those used for raw treatment, although the chloridized ore occupies more space than the raw. Solutions should be used consecutively, to find if fouling occurs, and solutions should be passed through zinc-boxes, as in the case of raw ores. When a preliminary wash is advisable, it should be continued until the soluble base-metal chlorides, as well as the excess of salt, are washed out. In such a case, precipitation in separate boxes is

best. Silver chlorides may be precipitated in the front compartment on shavings; if gold is present it may be precipitated subsequently on zinc shavings. At Hacienda Guadalupe, in treating a quartz-ore with few base metals, the water-wash was passed through ordinary zinc-boxes, but the product collected in these was base.

To secure correct results, the ore, both before and after chloridizing, should be weighed, to find if either increase or decrease in weight takes place. It is also necessary to find out what difference in weight the water-wash will occasion, by leaching out the base-metal chlorides.

Plant.

When the preliminary tests are sufficiently satisfactory to warrant the adoption of the process, the design for a commercial plant can be made.

The required number and size of leaching-tanks must be calculated, as also, in the case of slimes-treatment, the number and size of agitators. It is advisable, especially in the latter case, to provide an excess-capacity of at least 20 per cent., for the contingency that the time of treatment may have to be lengthened. The use of large leaching-tanks is to be deprecated, because difficulties occur (particularly in the case of chloridized ores) which require instant remedy; results are obtained and defects remedied more speedily on a small charge; and the loss through defective treatment or carelessness is confined to a smaller tonnage in a small than in a large tank. A convenient size of leaching-tank, holding about 20 tons, is 17.5 ft. in diameter and 4 ft. deep. For slimes, the size of tank used depends purely on convenience and local conditions, since accurate pulp- and solution-samples can be taken, and defects remedied during treatment. As regards material for tanks, painted iron or steel is undoubtedly the best. The tanks should be round and not rectangular. In the latter case the sides buckle and get out of shape. As mines, however, are often at some distance from the railroad, and connected with it sometimes not even by cart-roads but by dangerous trails necessitating mule-transport, the choice of material for tanks depends more on local circumstances than on suitability. A mule carries 300 lb., 150 lb. on each side of the pack;—and the smaller the bulk of the

load, the easier for transport. Any piece more than 4 ft. long is inconvenient; and although pieces of 10 ft. can be carried, they have to be paid for extra, and on narrow trails often get injured by striking obstacles. It will be easily seen that the use of iron or steel plates sometimes presents such difficulties as to both transport and construction, as to make the choice of a different tank-material imperative. Redwood tanks are manufactured in the United States, and are satisfactory if preserved from extremes of dryness and moisture. This is difficult in plants not covered, like those which, in most parts of Mexico, are exposed to the sun during a dry season of eight or nine months. Most tanks have a groove from 2 to 3 in. deep in the upper part of the stave, which, filled with water or solution, keeps the upper part of the tank watertight. This construction requires a good deal of looking after, and is not always effective, especially in the case of large slimes-plants. Redwood tanks are more easily transported than iron ones, and require less skill in constructing; but the difficulties of transport are serious, and unless the pieces are very carefully protected, they are liable to irreparable damage on the road.

In some cases, where mines are far from the railroad, the expense and the dangerous trails preclude the use of either steel or redwood tanks, and the choice is limited to the materials the country can supply.

Most of the highlands and sierras are covered with pine trees, and boards or logs can be obtained at from $12 to $20 per 1,000 ft. board-measure. In such cases, planks can be selected free from knots, and ordinary carpenters can build small tanks at reasonable expense, say from $40 for a tank 9 ft., to $80 for a tank 17.75 ft. in diameter—which is perhaps the largest size that can be built by unskilled labor.

Wooden tanks are preferable to those of cement or masonry, because there is always a doubt whether the latter are solution-tight; they are also liable to leaks, not only on account of defective construction, but by reason of cracks and disturbances caused by earthquakes.

Cases may occur where wood is scarce and lime and stone abundant, and where Portland cement can be freighted at low rates, due to its adaptability for mule-transport. In such cases masonry tanks with cement bottoms and sides can be used, pro-

vided the construction is carefully attended to, and the tanks are relined occasionally with a coating of fresh cement.

As a still cheaper form of tank, and one which would be less liable to injury by earthquakes, I suggest the use of masonry with puddled clay cores, and preferably with wood lining. Such tanks are impervious and do not leak if properly built, but they require a solid rock foundation.

Extractor boxes may be made of iron or redwood; if of the former, they must be painted every two or three weeks, so as to prevent the formation of a zinc-iron couple, and consequent precipitation of the gold on the iron. Boxes may be made of native pine, but as the expense of getting proper boxes is not serious, this is to be deprecated.

Where the cost of freighting is heavy, bottom discharge-doors, if used, should be designed of a suitable size for mule-transport, and the pipes calculated so as to use the smallest diameter consistent with efficiency, and cut in 10-ft. lengths (the maximum for mule-transport), threaded and protected at both ends.

Product-dryers can be replaced by small reverberatory furnaces, either closed in or with a hood, the latter being preferable.

Where agitation is necessary, compressed air can be used to advantage. Where transport forms an obstacle, air compressors can be dispensed with, and equally satisfactory agitation secured by centrifugal pumps.

The question of the choice of a suitable furnace for previous chloridizing roast is important. White-Howell and Brückner furnaces have been used with satisfactory results; but any kind of mechanical furnace may be barred, when the advantages secured do not compensate for the cost of transport. What has been said regarding the difficulties of transporting tanks to mines at a distance from the railroad, applies still more forcibly to the case of mechanical roasters, which have to be sectionalized, thus adding to the initial cost, and incurring the drawbacks of all sectionalized machinery. Where mechanical roasters have been found impracticable, or even in other cases, various forms of reverberatory furnaces have been used. The advantage of these is that a better roast can, as a rule, be obtained; but the cost of handling, and of repairs where bricks are not suitable for standing heat, make these furnaces very

expensive. A really satisfactory furnace, which could unite cheap handling, economy of fuel and a satisfactory means of recovering the loss in volatilized metals, is much required.

Commercial Treatment.

Chloridizing.—The question of crushing the ore before treatment deserves due consideration, and the choice lies between dry and wet crushing. Dry crushing has the disadvantage of dust and extra expense in wearing-parts, but yields a product which can at once be chloridized. Wet crushing, which is undoubtedly preferable, yields a wet product in sands and slimes, which requires drying so as to reduce the moisture to not more than about 10 per cent., before placing it in the roaster. The following are the costs of treatment in Mexican currency, in a White-Howell and a reverberatory of the usual type, and include the drying of sands and slimes.

Cost Per Ton of Chloridizing Roasting.

	White-Howell.	Reverberatory.
Superintendence,	$0.13	$0.50
Labor,	1.53	2.01
Fuel,	2.53 (0.42 cords)	3.25 (0.45 cords)
Salt,	0.93 (2 per cent.)	2.78 (5 per cent.)
Tools and repairs,	0.02	0.08
Various stores,	0.52	0.06
	$5.66 Mex. per ton.	$8.68 Mex. per ton.
At 50c. U. S. per $1 Mex.,	$2.83 U. S. "	$4.34 U. S. "

It is difficult to institute comparisons where conditions vary, but by reducing the percentage of salt in the case of the reverberatory to that of the White-Howell, the cost of chloridizing by the former would be $3.42 U. S. as compared with $2.83 U. S. by the latter.

The following is an example of the cost per ton of chloridizing in Brückner furnaces supplied with dry-crushed pulp: Superintendence, $0.06; labor, $0.44; fuel, $2.13 (0.27 cords); salt, $2.74; tools and repairs, $0.26; various stores, $0.18; total, $5.81 Mex., or, at 50c. U. S. per $1 Mex., $2.90 U. S. per ton.

Reducing the amount of salt used to the quantity given previously for the White-Howell, the above figures are reduced to $2 U. S. per ton. The saving effected is represented by a reduction of 0.15 cords of wood per ton, and of labor used in

drying the pulp and slimes, which costs from 40c. to 80c. U. S. per ton.

These costs are given merely as a rough indication. Local conditions and size of plant must necessarily influence them to a large extent. In a White-Howell plant, treating 80 tons per day with firewood at $3.50 U. S. per cord and using 3.5 per cent. of salt, costs were reduced to as low as $1.38 U. S. per ton.

Leaching: Example No. 1.—The plant in question has a capacity of 700 tons per month. The chloridized pulp is charged into filters 18 ft. in diameter and 4 ft. deep, and receives as preliminary treatment 4 hours' soak with water and 7 hours' percolation. This water-wash is passed through special zinc-boxes, where an impure precipitate containing gold and silver is formed, and a consumption of about 0.4 lb. of zinc per ton takes place. When the salt and impurities have been removed by the water-wash, the charge receives 15 hours' soak and 90 hours' percolation with cyanide solution of 0.3 per cent., or 6 lb. to the ton of water, the ore receiving an equivalent of 1.5 of solution to 1 of ore, or 0.45 per cent. To displace the strong solution, 12 hours of percolation with a 0.05 per cent. weak solution is given, in the proportion of 1 of water to 2 of ore, and a final 30 hours' percolation with water-wash. The consumption of cyanide is 2.15 lb. per ton, and of zinc 1.08 lb.

The extractions are: Gold, 76; silver, 85.25; total value, 81.77 per cent. It will be observed that the gold extraction is not as good as the silver, a fact often noted in chloridized ores. The following are the costs, which can be considered as typical in small plants, although in some instances they have been reduced: Superintendence, $0.62; labor, $0.96; cyanide, $1.71; zinc, $0.29; laboratory, $0.10; various stores, $0.50; total, $4.18 Mex., or, at 50c. U. S. per $1 Mex., $2.09 U. S. per ton.

Leaching: Example No. 2.—The plant is capable of treating 1,800 tons per month. The treatment, with small variations, is practically the same as in Example No. 1. The consumption of cyanide is 1.73 lb. and of zinc 1.27 lb. per ton, the high consumption of zinc being due to the ore averaging over 30 oz. of silver per ton. The extractions are: Gold, 82.30; silver, 77.32; total value, 79 per cent. The costs are: Superintendence, $0.33; labor, $0.31; cyanide, $0.99; zinc, $0.33; laboratory, $0.10;

various stores, $0.22; total, $2.28 Mex., or, at 50c. U. S. per $1 Mex., $1.14 U. S. per ton.

Slimes Treatment: Example.—The plant has a capacity of from 750 to 800 tons per month, and is equipped with 7 tanks, 28 ft. in diameter by 10 ft. deep, 1 intermediate solution-clarifying tank, 4 zinc-boxes 2 by 20 ft., 2 sumps 28 by 10 ft., 1 No. 3 Cameron general-service horizontal pump, and 1 Leyner 8.5 by 10 in. compressor. It should be noted that the capacity of the compressor (100 cu. ft. per min.) was found to be too small; and although satisfactory results were obtained, a compressor of double the capacity would have facilitated the work. The ore, which has a specific gravity of 3.7, and carries galena, blende, pyrite and chalcopyrite, is crushed in Bryan mills, sized by hydraulic separators and concentrated over Wilfley tables. The slimes, which are about one-third of the original ore, freed as much as possible from the heavier metals, flow through launders direct into the center of the cyanide-tank, where they settle, the water flowing over the sides of the tank being fairly clear. When the tank has been charged and contains from 50 to 60 tons of ore, from 200 to 300 lb. of lime are added, and the charge is allowed to settle. Settling and the decantation of excess-water takes from 10 to 12 hours. The depth of charge is then carefully measured, and samples for specific gravity and assay are taken. The appendix shows a working-chart made by Mr. Gordon Wilson for controlling the operations. The chart refers only to one tank, but can be taken as typical of the treatment. In this instance the depth of the charge was 60 in., the specific gravity of the ore was 1.375, and its content of moisture was 55.64 per cent.; under these conditions, the following values, in U. S. currency, were obtained: Gold (at $20.67 per oz.), $1.80; silver (at 50 c. per oz.), $7.54; total, $9.34.

Calculating each foot of tank to contain 19.21 short tons of water, the charge contains 58.5 tons of ore and 73.5 tons of water.

As soon as the samples are taken, the first strong solution in sufficient quantity to bring the charge to the proportion of two of water to one of ore, is added. This solution consists of weak sump-solution (carrying about 0.06 per cent. of cyanide), strengthened to 0.1 per cent. Reference to the chart will show that in

the tank taken as an example, 18 in., or 28.8 tons, of weak solution were added, and strengthened with 166 lb. of cyanide, thus bringing the solution up to 102.3 tons to 58.4 tons of ore. The charge then receives as complete agitation as the compressor will allow, agitation being given in this instance by an iron pipe connected with the main air-pipe by a rubber hose, the pipe being placed in different parts of the tank by a man whose business it is to see that the whole charge receives attention in turn. In the tank under consideration, the first solution was agitated for 20 hours, and then decanted, the decantation of 20 in. of solution taking 7 hours. The solution meanwhile has dropped to 0.05 per cent. of cyanide; it is then strengthened, and precisely the same operation is repeated, as with the first solution. As many solutions are thus added and decanted as the working of the plant will permit, in the instance under consideration the total number being 11. Only the first three solutions need strengthening; the ore apparently demands a certain amount of cyanide to be saturated, and no further, or hardly any further, consumption takes place, the sump-solution remaining at from 0.05 to 0.06 per cent. Before decanting a solution, lime is added for settling, as required, the total consumption amounting to between 30 and 33 lb. per ton of ore. It is worthy of note that the first two strong solutions only take about 12 per cent. of the values into solution, and that it is only after the ore has had sufficient cyanide to satisfy it, that the subsequent two or three solutions dissolve from 50 to 60 per cent. The subsequent solutions then gradually decrease in effect, until finally they act merely as washes, the last solution carrying from 1 to 2 per cent. of the values.

The tank under consideration was 49 hours charging, 11 hours decanting excess-water, 21 hours pumping, 78 hours decanting, 175 hours agitating, total 13 days and 22 hours. The assays of tails showed: Gold, $0.05; silver, $1.13; total, $1.18; or an extraction of gold, 97.22; silver, 85.01; extraction of total value, 87.36 per cent.

The difference in assay-value between washed and unwashed tailings was only $0.08, proving that the washing was complete. The total cyanide-consumption was 5.18 lb. per ton, of which about 4 lb. per ton was chemical, and the remainder mechanical consumption, 53.6 lb. of cyanide running to waste with the

ultimate 44.7 tons of solution. This mechanical loss could be remedied to a certain extent by a greater sump-accommodation.

The costs were as follows:

Labor: Superintendence,	$0.4068	
Pumping,	0.0690	
Various,	0.0829	
Discharging,	0.1508	
Stores:		
Cyanide,	3.5313	(5.18 lb. per ton.)
Zinc,	0.2233	(0.50 lb. per ton.)
Lime,	0.2268	(33.17 lb. per ton.)
Light,	0.0440	
Various,	0.0130	
Laboratory,	0.2947	
Shops,	0.0379	
Total,	$5.0805 Mex.	
Or at $0.50 U. S. per $1 Mex.,	$2.5402 U. S. per ton.	

It will be seen that by far the greatest expense is for cyanide, $1.7657 U. S. per ton, leaving $0.7745 per ton for the rest of the treatment. This latter sum, under favorable circumstances, could be very considerably reduced in a large plant.

Appendix.

Working Chart of Operations in Tank No. 2. Slimes-Treatment, Example (see page 28).

Treatment.	Bulk of Solutions.				Started.		Finished.		Strength.		
	Before.	After.	Drawn.	Drawn.	Date.	Hour.	Date.	Hour.	Date.	KCy.	Strength. Per Cent.
	In.	In.	In.	Tons.							
Charging	.....	.....	.....	.....	May 10	8 p.m.	May 12	9 p.m.	...	.....	.....
Decanting water	.....	.....	.....	.....	" 12	9 "	" 18	8 a.m.	...	.....	.....
Pumping first solution	.....	.....	.....	.....	" 13	10 a.m.	" 13	11 "	13	166 lb.	.....
Decanting No. 1	78	58	20	32.0	" 14	7 "	" 14	2 p.m.	14	.....	0.05
Pumping No. 2	.....	.....	.....	.....	" 14	3 p.m.	" 14	4 "	14	82 lb.	.....
Decanting No. 2	73	56	17	27.2	" 15	10 "	" 16	6 a.m.	15	.....	0.07
Pumping No. 3	.....	.....	.....	.....	" 16	10 "	" 16	11 p.m.	16	55 lb.	.....
Decanting No. 3	71	53	18	28.8	" 18	7 a.m.	" 18	11 "	...	.....	.....
Pumping No. 4	.....	.....	.....	.....	" 19	2 a.m.	" 19	4 a.m.	...	.....	.....
Decanting No. 4	71	53	19	30.4	" 19	5 p.m.	" 19	10 p.m.	...	.....	.....
Pumping No. 5	.....	.....	.....	.....	" 19	11 "	" 20	2 a.m.	...	.....	.....
Decanting No. 5	73	55	18	28.8	" 20	4 "	" 20	9 p.m.	...	.....	.....
Pumping No. 6	.....	.....	.....	.....	" 20	9 "	" 20	11 "	...	.....	.....
Decanting No. 6	72	57	15	24.0	" 21	4 a.m.	" 21	10 a.m.	...	.....	.....
Pumping No. 7	.....	.....	.....	.....	" 21	10 "	" 21	12 m.	...	.....	.....
Decanting No. 7	74	58	16	25.6	" 21	2 p.m.	" 21	7 p.m.	...	.....	.....
Pumping No. 8	.....	.....	.....	.....	" 21	7 "	" 21	9 "	...	.....	.....
Decanting No. 8	73	60	13	20.8	" 22	8 "	" 22	12 "	...	.....	.....
Pumping No. 9	.....	.....	.....	.....	" 23	1 a.m.	" 23	4 a.m.	...	.....	.....
Decanting No. 9	82	60	22	35.2	" 23	7 "	" 23	2 p.m.	...	.....	.....
Pumping No. 10	.....	.....	.....	.....	" 23	2 p.m.	" 23	4 "	...	.....	.....
Decanting No. 10	78	66	12	19.2	" 23	7 "	" 23	11 "	...	.....	.....
Pumping No. 11	.....	.....	.....	.....	" 23	11 "	" 24	1 a.m.	...	.....	.....
Decanting No. 11	90	60	30	48.2	" 24	7 a.m.	" 24	6 p.m.	...	.....	.....

	Gold. $20.67 Per Oz.	Silver. $0.50 Per Oz.	Value of Total. U. S.
Heads	$1.80	$7.54	$9.34
Tails	0.05	1.13	1.18
Extraction	1.75	6.41	8.16
Per cent. extracted	97.22	85.01	87.36

Difference between washed and unwashed, . . $0.08
Cyanide-consumption, ... 5.18 lb.
Extraction by solutions, ... 80 per cent.
Depth of charge, ... 60 in.
Ore, ... 58.4 tons.
Specific gravity, ... 1.37.
Moisture, ... 55.64 per cent.

A Bituminous-Coal Breaker.

BY LEWIS STOCKETT, STOCKETT, MONT.

(Atlantic City Meeting, February, 1904.)

DURING the year 1903, a building having machinery corresponding to an anthracite-coal breaker was erected at the town of Stockett, Cascade county, Mont., for the purpose of breaking-up and cleaning bituminous coal.

The coal is mined from a vein of the Kootenai group of the Lower Cretaceous measures having the following section:

	Feet.	Inches.	Feet.	Inches.
Sandstone ledge				
Top coal	1	2		
Slate			0	5
Top bench	1	6		
Bone			0	7
Gray coal	1	3		
Blacksmith coal	2	5		
Bone			0	6
Coking coal	1	5		
Metamorphosed fire clay				
Total — Coal	7	9		
Total — Refuse			1	6
			9	3

The vein is nearly horizontal in position, and each bench differs in quality as shown in Table I.

TABLE I.—*Analysis and Specific Gravity of Stockett Coals.*

	Moisture.	Volatiles.	Fixed Carbon.	Ash.	Sulphur.	Specific Gravity.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
Top coal.....	4.35	30.90	54.90	9.85	2.56	1.29
Top bench.....	2.85	28.95	53.45	14.75	1.64	1.29
Gray coal.....	3.05	25.10	51.55	20.30	1.33	1.50
Blacksmith coal.	2.65	35.85	51.35	10.15	1.45	1.29
Coking coal.....	3.01	29.55	52.09	15.35	1.92	1.44

There are no partings whatever between the different benches of coal, slate and bone, the vein being very hard and a solid mass from top to bottom. This, with a liberal number of pyrite-nodules, called sulphur-balls, scattered promiscuously throughout the vein, makes the coal a difficult one to clean. Previous to the building of the breaker, the cleaning was done with indifferent results by the miner at the face, and the nut-coal was washed in a jig. Scarcity of water, however, prevented the erection of a plant to wash the whole product, and at times necessitated the shutting down of the nut-coal washer.

The difficulties to be overcome in separating the different materials of the vein will be appreciated by a study of their specific gravities, shown in Table II.

TABLE II.—*Specific Gravity of Material Mined at Stockett.*

Sulphur balls, . . .	4.140	Gray coal, . . .	1.5 to 1.751
Slate, ...	2.402	Coking coal, . . .	1.440
Bone, ...	1.963	Other coal, . . .	1.290

After a series of experiments, a scheme of treatment supplementary to the cleaning by the miner was determined upon and carried out, one of the requirements being so to connect the new work to the old tipple that the change could be made from one to the other without delay in the shipment of coal. This arrangement was successfully accomplished and accounts for some things about the building that may be open to criticism,—things which would be arranged differently in an entirely new plant. Ground was broken for the foundations of the building, March 26, 1903, the timber-work was commenced April 16, the machinery was started August 10 and has since been in continuous operation every week-day with gratifying results.

The following illustrations have been prepared from the working-drawings to show the general arrangement of the plant and the manner in which the machinery has been installed:—Fig. 1. Ground plan; Fig. 2. Longitudinal section showing especially the machinery; Fig. 3. Transverse section showing the machinery; Fig. 4. Transverse section showing the bins; Fig. 5. Plan at the screens; and Fig. 6. Section of the tipple. The letters in the text of this paper, referring to the legends, are common to all of the illustrations.

The coal, carried from the mine in pit-cars of a capacity averaging about 1.5 tons to the car, is weighed on an automatic scale and thence dumped by the cross-over tipple *A*, over the bar-screen *B*, 12 ft. long, 6 ft. wide and having a pitch of 6 in. to the foot with spaces between the bars 2 in. wide, which screens out that portion already small enough (about 30 per cent. of the total). It then falls on the shaking-screen *C*, having 1-in. round perforations, 6-in. throw and 100 strokes per minute, which removes the slack from the coal. The slack is loaded directly into the railroad-car or taken to the boiler-room by means of a wire-rope conveyor. The portion of the coal that goes over the shaking-screen *C* slides into a hopper *D*, from which it feeds into an elevator *E*, consisting of a rubber belt 16 in. wide, having 8- by 14-in. buckets placed every 16 in. and operated with a speed of 225 buckets per minute. This belt elevates the material to the top of the building.

The coal that passes over the screen *B* falls upon a traveling belt *F*, 4 ft. 6 in. wide and 26 ft. long, having a speed of 33 ft. per minute; and from this belt any large pieces of slate or other impurities, machine-picks, car-couplings, sprags, etc., are removed by men stationed on the sides and thrown into a rock-elevator *G*. This traveling belt is operated by a clutch gear *H*, which in case of a very large quantity of impurities appearing is thrown out, the belt stopped and all of the impurities removed before the coal drops into the rollers *I*, which reduce the coal to a size not exceeeding a 4-in. cube. It was found necessary to reduce to a 4-in. size in order to prevent the concealment of a sulphur-ball in a lump of coal. The rollers are of the removable-tooth style, 36 in. in diameter and 48 in. wide, and revolve at 75 revolutions per minute.

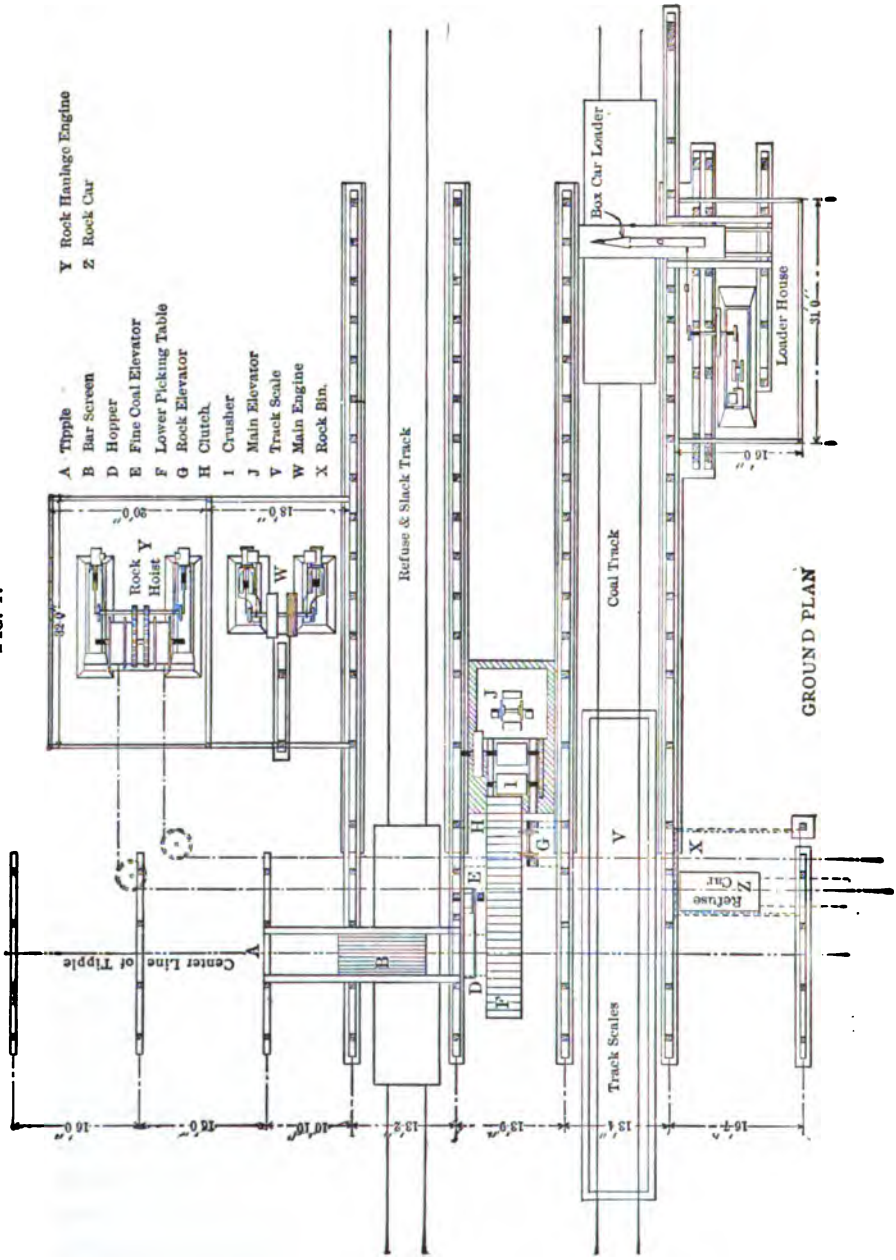

FIG. 1.

BITUMINOUS-COAL BREAKER OF THE COTTONWOOD COAL CO., STOCKETT, MONT.

From the rollers the coal is elevated by a continuous elevator *J*, having buckets 12 by 30 in., and operated at a speed of 65 buckets per minute. Each bucket has a capacity of 110 lb. of

coal when level-full, which is equivalent to 200 tons per hour; the capacity of the fine-coal elevator *E* is 90 tons per hour, giving a combined elevating-capacity of 290 tons per hour, or

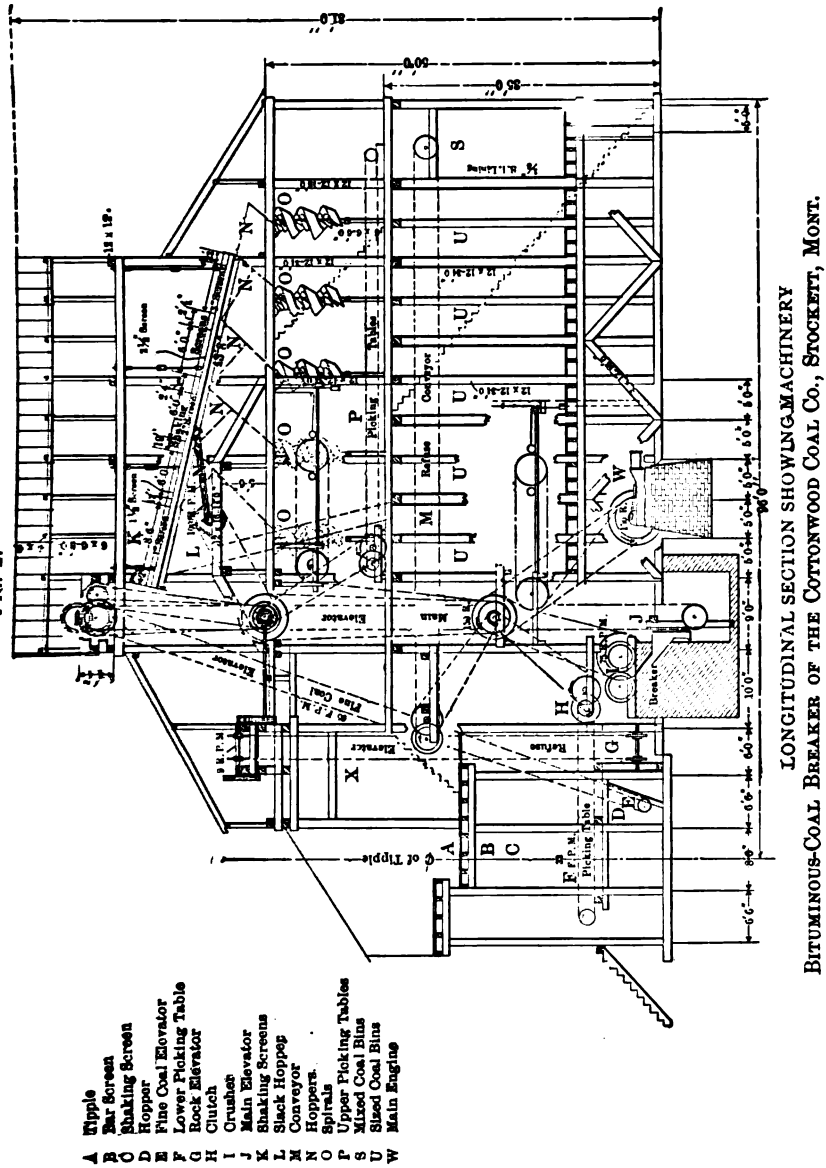

FIG. 2.

LONGITUDINAL SECTION SHOWING MACHINERY

BITUMINOUS-COAL BREAKER OF THE COTTONWOOD COAL CO., STOCKETT, MONT.

2,900 tons per day of 10 hours, an amount which added to the slack screened out at *C* gives a total capacity of 3,200 tons per day.

The coal elevated by the elevators, *E* and *J*, is evenly divided over the shaking-screens *K*, 5 ft. wide and 46 ft. long, having a 3-in. pitch to the foot, a 6-in. throw and 100 strokes per min-

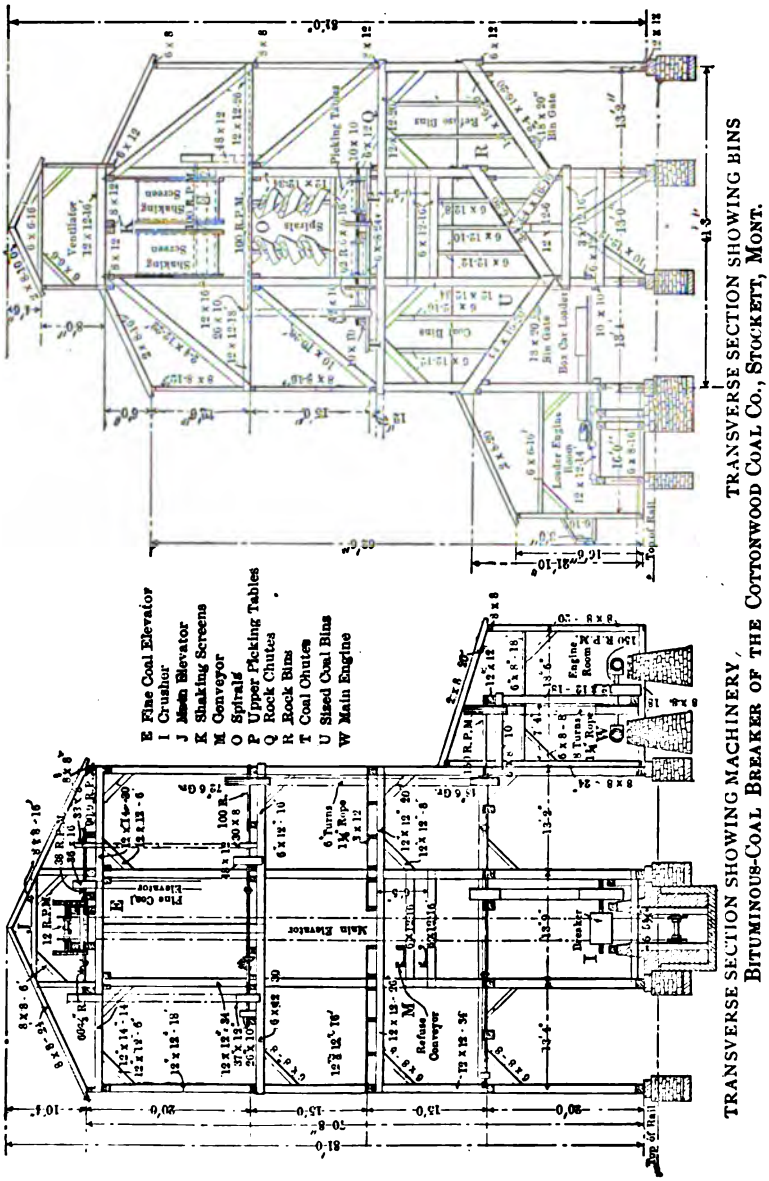

FIG. 3. TRANSVERSE SECTION SHOWING MACHINERY, BITUMINOUS-COAL BREAKER OF THE COTTONWOOD COAL CO., STOCKETT, MONT.

FIG. 4. TRANSVERSE SECTION SHOWING BINS

ute. The plates of the screens have respectively 1-, 1.5-, 2-, 2.5- and 3-in. round perforations, and separate the coal into slack, pea-, nut-, stove-, egg- and broken-sizes.

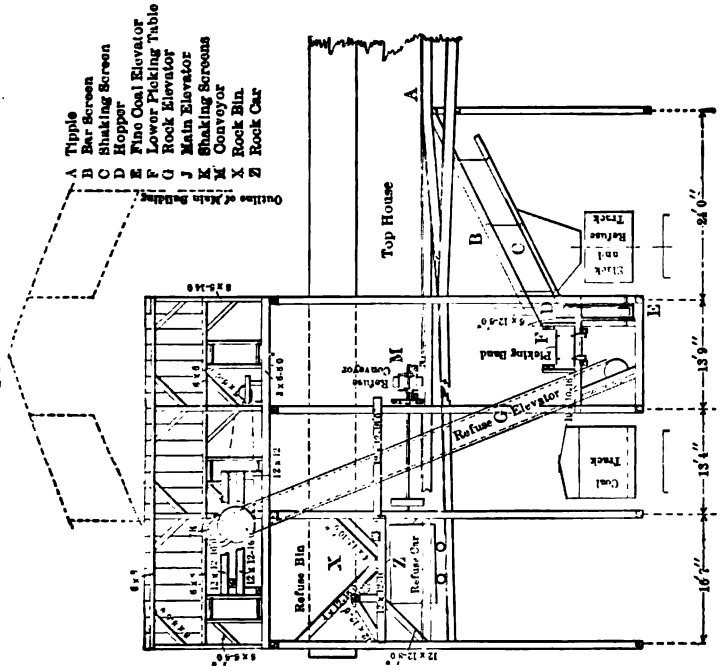

FIG. 6. SECTION OF TIPPLE

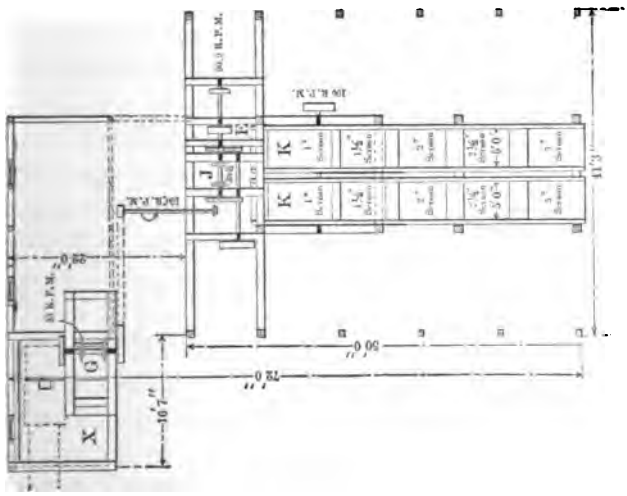

FIG. 5. PLAN AT SCREENS

BITUMINOUS-COAL BREAKER OF THE COTTONWOOD COAL CO., STOCKETT, MONT.

The slack resulting from the breakage of the coal is found to be clean, and, not needing any further preparation, it descends through the hopper *L* to the top strand of the conveyor *M*

(having 10- by 20-in. flights spaced every 3 ft.), and taken directly to the mixed-coal bin *S*. The other sizes are fed by means of the hoppers *N* into spiral separators *O*, which separate the greater part of the impurities from the coal. These impurities pass either to the lower strand of the conveyor *M* and are conveyed to the rock elevator *G*, or from one set of spirals to the bins *R*, by means of chutes *Q*, which give an opportunity to re-pick the refuse by hand and save any coal that may be in it. This refuse is finally loaded into railroad-cars and used by the railroad for widening banks, etc.

From the spirals *O* the coal drops on to two picking bands *P*, 4 ft. wide and 50 ft. long and having a speed of 40 ft. per min., which convey it to the mixed-coal bin *S* and give an opportunity to pick out by hand any impurities not removed in the spirals; from one set of spirals, inclined chutes *T* pass the coal into bins *U* for loading straight sizes, any remaining impurities being removed by hand while the material is on these chutes.

The rock elevator *G*, having continuous buckets 12 by 30 in., and speed of 50 buckets per minute, elevates the impurities into a bin *X*, from which it is loaded into a 6-ton car and hoisted by a pair of geared tail-rope engines, with 10- by 18-in. cylinders, to the top of the adjoining hill and automatically dumped.

The machinery of the entire plant is driven by a double engine *W*, with cylinders 13 by 18 in., running 150 rev. per min.; the connection to the first and second line-shafts being made by rope drives, and all other connections by rubber belts.

From the bins *S* and *U* the coal is loaded into railroad-cars, a box-car loader being placed opposite the chute *S* to load box-cars. The cars are weighed on the scale *V*, and it may be of interest to know that, when loading the mixed-coal into box-cars, the entire product of the breaker passes through an opening 12 by 14 in. in area.

The building is heated by steam-coils supplied with live steam, and is lighted by incandescent electric lights. It has two stand-pipes with hose-connections and hose for fire-protection.

On account of the slight difference in the specific gravity of the gray coal and the bone, the spirals are adjusted so as to retain only the slate and flat sulphur balls, leaving the bone to be removed by hand. The round sulphur balls which on account of their shape are the first to leave the spirals and go with the coal, also have to be removed by hand.

The results obtained are given below, showing the percentages of refuse in the various sizes:—In pea coal, 4; in nut coal, 3; in stove coal, 3; in egg coal, 2; in broken coal, 1; and in mixed coal, 2.5 per cent. Of 2,000 tons of the mine-product, which is daily dumped into the breaker, 200 tons of the various impurities are removed, and these impurities do not contain on an average over 1 per cent. of coal.

The mixed coal is used by the railroads as a locomotive-fuel, and proves an excellent article; the various sizes are used in the commercial trade, the slack- and pea-sizes making the very best of boiler-fuel.

The cost of the breaker, in the section where the highest wages are paid in the United States and where freight is a very large item, was $42,517.90, divided as follows:

General expense,	$472.45	Scale,	$123.75
Foundations,	2,445.46	Hardware,	805.91
Lumber,	3,607.31	Labor,	10,700.82
Machinery,	12,764.89	Freight,	8,844.13
Separators,	2,753.18	Total,	$42,517.90

The above includes the cost of 300,000 ft. of lumber and 400 yards of masonry as well as of duplicate pieces for all of the parts of the machinery that are liable to break or wear out.

The cost of operation of the plant over and above that of the force previously used on the old tipple is as follows:

Coal inspector on lower picking band,	$3.00	Cost per ton of 1,500 tons (1,800 less 300 slack),	0.03
3 men on lower picking band,	7.50	To which add interest on investment, taxes, insurance, wear and tear, supplies used and repairs, which as near as can be estimated is per ton,	0.03
1 man looking after screens,	2.50		
1 man looking after spirals,	3.00		
15 boys picking slate, etc.,	15.00		
1 breaker boss,	3.00		
1 engineer, also hoists the rock,	4.00	Total additional cost per ton,	0.06
1 machinist, oils and keeps machinery in order,	3.50		
1 man loading rock,	2.50		
	$44.00		

The largest item of cost per ton, however, comes from the decrease of production from what was formerly shipped, by reason of the removal of the impurities; the total cost being now charged to 1,500 tons of lump-coal instead of 1,700 tons, as for-

merly; this is met by the increased price received for the coal, which at the increased price is a cheaper fuel to the consumer than the former product at the lower price.

The success of this plant will make available large fields of coal in Montana, which on account of the impurities present were hitherto regarded as unworkable. However, it is only with coals hard enough to keep their shape, and where a marked difference exists both in the specific gravity and the shape of the coal and the impurities, that the spirals will work successfully.

Note on the Relation Between Arsenic and Electro-Motive Force in Copper-Electrolysis.*

BY L. WEBSTER WICKES, ANACONDA, MONTANA.

(Atlantic City Meeting, February, 1904.)

THE bad effects of arsenic in commercial copper are well-known to metallurgists.

The refining of copper by electrolysis eliminates practically all of the arsenic, provided certain requirements are fulfilled. The first, and most important, of these requirements is, that the voltage of the depositing current shall not be too high. It has been known for some time that the percentage of arsenic in the deposited copper increases with the voltage.

To investigate the relation between the electro-motive force (e.m.f.) and the freedom of the precipitated copper from arsenic, the experiments were made, of which a brief summary is here given:

I made three copper-arsenic alloys, containing respectively 3.56, 2.24 and 0.96 per cent. of arsenic. From these alloys I cast several anodes. For cathodes I used sheets of thin copper,

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* Abstract of a thesis presented, in partial fulfillment of requirements for the degree of Metallurgical Engineer, to the Faculty of Applied Science, Columbia University, N. Y. City, May, 1903. The full text of this thesis, with tables, curves, etc., is in the library of the Department of Metallurgy at the University, and may be consulted by those desiring more detailed information.

coated on one side with "mounting wax," and on the other with a thin film of graphite. Vaseline was used to make the graphite adhere. During electrolysis the copper was prevented from depositing on one side of the cathode by the wax, while the copper deposited on the other side was readily stripped off, to be weighed, analyzed, etc. The electrolyte used was that suggested in Peters's *Modern Copper Smelting*, viz.: H_2O: $CuSO_4$: H_2SO_4=75: 19: 6. The electrodes were 6 by 8 in., resting in 5-liter vulcanite vats. Electrolytes were circulated by air.

I set up three sets of three vats, each connected in series, and each containing an anode from each of the three alloys. The current used in each set of three cells was so regulated that the cell-drop of the first set was 0.8 volts; of the second, 0.6 volts; and of the third set, 0.4 volts.

The amperes of the first set were 5.5, and of the second and third were 4. The difference in voltage of the second and third was obtained by decreasing the distance between electrodes.

The question was raised, whether the higher voltage, due to the greater distance between the electrodes, would have the same effect as increasing the voltage by increasing the current, and keeping the cell-resistance constant. Undoubtedly increasing the e.m.f. by increasing the current changes the drop from anode to solution, and from solution to cathode, as well as through the solution. But, since the energy set free by the solution of a metal is equivalent to that required to precipitate from the solution, they will always balance each other.

I found that, with different voltages, but with the same percentage of arsenic in the anode, the percentage of arsenic in the cathode per weight of copper is practically the same, whether the difference of voltage is due to difference in the distance between the plates, or to difference in amperage.

I set up four more vats, using the 2.24 per cent. arsenic alloy for anode in each. To the electrolyte of each was added 0.101 per cent. As in the form As_2O_3. The first of these cells had 2.5 in., and the second 1.5 in. between the plates; and passing 4.5 amperes through the first, and 9.5 amperes through the second, gave them an equal cell-drop of 0.95 volts. The electrolyte was the same in both. In cell No. 3 a less acid electrolyte was used than in cell No. 4 (other conditions being the

same as in Nos. 1 and 2), its conductivity being brought up to that of No. 4 by the addition of potassium-sulphate. The cell-drop of each was 0.95 volts when No. 3 had 2.5 in. between plates and 4.5 amperes, and No. 4 had 1.5 in. between plates and 9.5 amperes. No. 3 was kept quite warm.

Two important facts were noted:

1. Whenever the main current was shut off, a certain voltage was always shown in the cell.

2. This voltage was always in the same direction as the original current.

Of the four cells last described, the percentage of arsenic per weight of copper was less in No. 3 than in No. 1, and less in No. 1 than in No. 2 and No. 4.

Since a certain voltage was shown in the cells when the main current was interrupted, said voltage being in the same direction as the main current voltage, it is evident that, as the electrodes are identical with the exception of the arsenic, the arsenic passed into solution as cathions, or out of solution as anions. In either case the result is the same. It is impossible for anions to exist on the cathode. Therefore, to produce this voltage, the arsenic must pass into solution as metallic arsenic, and from the solution it becomes sulphate. To be deposited on the cathode, it must continue to exist as cathion. In this water solution the arsenic hydrolyzes and becomes arsenate. When hydrolyzed it cannot be deposited. Hydrolyzation is a function of the time the ion is in contact with the solution. In other words, the more time the arsenic has to hydrolyze, the smaller will be the per cent. of arsenic in the deposited copper. As the amperes increase, the time for hydrolyzation decreases. The hydrolyzation in cell No. 3 was greater than that in No. 1 (less acid and warmer), while both gave greater length of time than Nos. 2 and 4.

From the above, I think it follows that the percentage of arsenic in the cathode is only indirectly a function of the voltage in so far as that is a function of the amperage, and is directly a function of the degree of hydrolyzation of the sulphate. The investigation of the exact limit to which the hydrolyzation of the arsenic sulphate could be carried without the production of the copper oxide, I was forced to abandon for the time being.

The arsenic was added to the second group, consisting of the four cells, to bring the electrolyte up to the condition it would be in after a considerable run. The arsenic in the successive cathodes was not increased thereby, as is the case with electrolytes high in arsenic, due to a long run. This fact would seem to indicate that the arsenic in the solution, after a run, was not in the form of As_2O_3. Mr. W. McA. Johnson, in one of his papers,[1] says that "The oxide of arsenic present in an anode is an insulator and as such passes into the slimes." The arsenic seems, therefore, to have another "phase." So far, this has not been investigated.

Another question which I have so far been unable to answer is, What effect has the condition in which the arsenic exists in the anode on its distribution throughout the cathode, electrolyte and slimes, or its combination with silver, etc.?

A careful research in this field, as well as along the line of the form or phase of the arsenic during the different stages, will be necessary before definite conclusions can be reached.

The opinions here brought forward are based on the analysis of nearly 200 depositions of cathode copper, and electrolytes.

The experiments were carried on in the Department of Metallurgy of Columbia University, to which I am greatly indebted for the opportunities offered. I am also indebted to Professors Morgan and Miller, of the Department of Chemistry, for much valuable help and many suggestions.

[1] *Transactions of the American Electrochemical Society*, vol. ii., p. 173.

Estimated Costs of Mining and Coking and Relative Commercial Returns from Operating in the Connellsville and Walston-Reynoldsville Districts, Pennsylvania.

BY EDWARD V. D'INVILLIERS, PHILADELPHIA, PA.

(Atlantic City Meeting, February, 1904.)

In connection with some recent professional work in the coalfields of Western Pennsylvania, with special reference to results of coking operations, I was asked to compile a statement giving the estimated cost of mining coal and manufacturing coke at two hypothetical plants of 500 ovens each, giving the relative yield of coal in coke at each plant; and, upon the average number of working days in each of the past five years, from 1899 to 1903, together with average selling price of coke at the ovens, to prepare an estimate of the relative commercial returns in each district.

What seemed at first sight to be a comparatively simple proposition, involving only the ordinary elements of mining and coking costs, became complicated by the lack of standard published records of mining costs and sales, which would be applicable to the subject at issue.

Upon searching through the usual channels of Government publications and the files of our technical societies and mining periodicals, I was surprised to note the absence of fundamental factors that would approximately meet the case in point.

The very elaborate and interesting statistics annually published by the National Government[1] do not supply the required data for the following reasons:—1. Being reported by arbitrary districts, the average values are too general to be applied to special fields. 2. The value of coal charged to the ovens is apparently based upon the regional average value of the coal if sold in the market, and not upon the cost of production. 3. Some plants only use the slack (a by-

[1] *Mineral Resources*, U. S. Geological Survey, Washington, D. C.

product) for coking while others use run-of-mine coal. 4. In the returns by operators, the value is sometimes based on washed coal and sometimes on the unwashed product of the mines, and finally, 5. The question of costs of coal and coke do not enter at all into the compilation of returns. Hence, the value of the coal charged to the ovens is, in a great many cases, an arbitary one and does not represent the actual cost of coal at all.[2]

Extending my inquiry further and availing myself of the generous friendship and esprit du corps of the mining profession, I found equal difficulty in harmonizing data kindly furnished,[3] for the reason that no man seemed to be able to recall what happened five years ago.[4]

Mine-superintendents and sales-agents were equally negative and indefinite in their statements, and I have found if I had to harmonize the figures furnished by representatives of these two departments of a commercial operation it would be difficult for me to explain why some people were in business at all.

Disavowing the slightest claim to either originality or exactness in the compilation offered, I wish to submit to the *Institute* a summary of the results obtained and the factors upon which they were based. I wish also—

1. To invite frank criticism and obtain exact data which will be serviceable in the future in connection with similar or parallel cases in other regions.

2. To enlist the co-operation of those who are actually engaged in mining and coking towards accumulating facts which are of the highest importance if made accessible; and

3. To induce a wider publication of actual figures used in compiling average costs for a period of years and relating to all of the principal mining regions, whereby a more definite conclusion may be drawn to govern and safeguard the investment of capital in new or less known districts.

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[2] For instance, the reports for 1901 and 1902 give the average price per ton for coal in Fayette county as $1; manifestly no such figure could be used in computing the actual cost of coal to the ovens in the Connellsville region for those years.

[3] Among many, I especially mention and thank Messrs. E. W. Parker, Washington, D. C.; E. F. Saxman, Latrobe, Pa.; E. J. Taylor and G. S. Baton, Pittsburg, Pa., for their valuable contributions.

[4] The man would indeed be a genius who would average the selling price of coke during the last half of 1902, when prices soared up from $4 to $10 per ton, and even higher for quick delivery.

The two plants assigned for consideration were:

1. A plant in Lower Connellsville, Fayette county—a slope mine—working the "Connellsville" or Pittsburg seam, averaging 7 ft. of mining coal on a pitch of about 6 per cent. and on land valued and capitalized at $750 per acre.

2. A plant in Reynoldsville, Jefferson county—a shaft operation—mining the "Reynoldsville" or Lower Freeport seam, averaging 6 ft. of mining coal, with a dip of about 2.5 per cent. in favor of the load and with land valued at $250 per acre.

Of course, both plants were assumed to be similarly equipped; with a capacity of 1,500 tons a day; with the same number of ovens and equal developments. Producing the same tonnage, what should each plant have earned during the past five years, assuming all the output to have been converted into coke and sold at regional prices?

As is the case in many other branches of a mining and manufacturing business, the cost of coal delivered to an oven, and the cost of the manufactured product, depends largely upon individual judgment or practice and to general management. Therefore, without having access to the accounts of a number of individual mines, it is not possible to do more than to approximate the average regional cost of mining coal or manufacturing coke.

Moreover, it is very difficult to take any published statistics bearing upon any one mining region and strike an average for cost of either mining the coal or producing the coke at any particular mine. For though each plant in a district may be mining upon the same scale of wages, the computation of net mining-costs may differ to a considerable extent in two adjoining plants; due to different methods of book-keeping, to a difference of opinion as to what items are properly chargeable to mining account and to capital account, or to physical differences at the two mines.

Then, again, costs and profits will vary somewhat in the same district where two mines are operated under different conditions—one a shaft-mine, with requirements for pumping and ventilation; the other a slope or drift-mine, with these items reduced to a minimum.

In the special cases under review the only point of close similarity is the size or mining-thickness of the coal to be

mined; and yet even here there are marked differences in the conditions to be met with.

The Connellsville[5] plant will mine the Pittsburg or Connellsville seam—the most persistent, most uniform and regular seam of coal in the whole Appalachian coal-field of the United States.

It is a coal bed of the highest integrity and the most free from integral impurity, having but two scarcely discernible thin slate bands, with a few inches of coal between, which portion of the seam is used for "bearing-in," or mining the coal.

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[5] In this paper the term "Connellsville" is used to signify "Lower Connellsville," though I protest against the arbitrary and unreasonable significance of the sub-divisions of the Connellsville district.

The *Connellsville Courier* gives its reasons for this ruling as follows, under date of January 21, 1904:

"The Upper Connellsville field is Connellsville coke only in a general geological sense; it is Connellsville coal in name, but not in fact. By the same token, the Lower Connellsville coke is made from Pittsburg coal, but in characteristics it is Connellsville coke."

We all remember when the only "pure Connellsville coke" was made at Connellsville, and how later it was found convenient to use this trade-mark on the product of Mount Pleasant and then of Pleasant Unity.

But the term was never applied north beyond the Pennsylvania railroad in the Connellsville basin: this field was named the "Upper Connellsville," and subjected to discrimination in sales and discrimination in wage scale.

Later, in 1900, it was found possible to make "Connellsville coke" to the south of Uniontown and even go outside of the basin entirely and create a new district—the Lower Connellsville—to incorporate the operations successfully making and selling coke from Perryopolis on the north to the Klondike area on the south, as well as south of Uniontown.

The strange result of this partition of the field is that in the Connellsville and Lower Connellsville districts the same wage scale prevails, though the seam mined is different physically and structurally, and in the Lower Connellsville field bears a most striking resemblance to the coal immediately north of the Pennsylvania railroad, in the Upper Connellsville district.

The mines in the latter field, however, pay a totally different and higher wage scale, equivalent to at least 10c. per gross ton more in cost of coal, while mining the same section as along the Redstone and more truly Connellsville coal.

As an evidence of the difference in wage scale existing in the Connellsville Region (and this in large measure vitally affects the cost of coke), the following table is arranged to show the rates paid by the H. C. Frick Co. and by a plant in the Upper Connellsville field, since December 16, 1903.

TABLE I.—*Rate of Wages Paid Since December* 16, 1903.

	Connellsville District.	Upper Connellsville District.
Mining and Loading:		
Room- and rib-coal,	$1.10 per 100 bu.	$0.45 per gr. ton.
Heading-coal,	1.25 per 100 bu.	0.52 per gr. ton.
Wet heading-coal,	1.30 per 100 bu.	0.52 per gr. ton.
Drawing coke per 100 bu. charged,	0.63	1.00 per oven.
Leveling, per oven,	0.105	0.09
Drivers and rope-riders (shafts and slopes),	2.20 per full run of 9 hours.	0.25 per hour.
Drivers and rope-riders (drifts),	2.10 full run.	0.25 per hour.
Cagers,	2.20 per full run of 9 hours.	0.25 per hour.
Wages per day:		
Tracklayers, blasters and timbermen (shafts and slopes),	2.20	1.60 to 2.40
Tracklayers, blasters and timbermen (drifts),	2.10	1.60 to 2.40
Assistant tracklayers and inside laborers,	1.65	1.60 to 2.40
Dumpers and tipplemen,	1.68	1.70
Teamsters,	1.60	1.70
Carters,	1.50	1.70
Yard laborers,	1.40	1.70
Chargers, per oven,	0.04	0.025
Chargers, per day,	1.75	
Trapper boys, per day,	0.93	1.25
Forking cars, 40,000 lb. capacity and less, per car,	1.40	1.50 to 1.60
Forking cars, 50,000 lb. and 60,000 lb. capacity, per car,	1.50	1.50 to 1.60
Forking cars, over 60,000 lb. capacity, per car,	1.60	1.50 to 1.60

The Reynoldsville plant will mine the Lower Freeport or Reynoldsville seam, certainly the best in its region, and perhaps of better quality and better section at and around Reynoldsville in Jefferson county than anywhere else in that field. But it is essentially a seam of variable thickness, due to a rolling roof or floor, and nearly always carries one or more streaks of bone-coal or impurity. Otherwise it is perhaps as pure a seam chemically as the Connellsville seam, except that it usually contains more sulphur as pyrite in parts of the field, which necessitates washing the coal before coking it.

In both regions it will be found desirable, if not necessary, to crush the raw coal before coking, although over a given screen, of say 1¼-in. mesh, there will probably be a greater amount of

coal to be crushed at Reynoldsville than in the Connellsville district.

The next point of difference will be in the yield per acre. The Connellsville mine will certainly average 7 ft. of mining-coal after leaving some coal in the floor and roof for better protection of the mine. The Reynoldsville mine will hardly yield an average of over 6 ft., so that there is at once a theoretical difference in content per acre of 1,600 tons in favor of the Connellsville district.

In good mining practice in the Connellsville field there is recovered from 80 to 85 per cent. of the estimated content, so that a 7-ft. seam there should furnish about 9,240 tons per acre, or 82.5 per cent.

In Jefferson county the recovery will not average more than 80 per cent., owing to inferior conditions for economical mining, so that a seam averaging 6 ft. there would not return over 7,600 tons per acre.

As a safe factor for estimating the actual returns in the two regions—providing for contingencies of variation in thickness, rock faults and troubles—it would be better perhaps to place the Connellsville yield at 9,000 tons and the Reynoldsville output at 7,200 tons per acre, figuring the recovery in the first instance at 1,300 tons per foot per acre or about 82 per cent., and in the latter at 1,200 tons or about 80 per cent.

This difference in tonnage yield, however, will be offset by the great difference in first cost of the land, the Connellsville coal-land being capitalized at about three times the amount invested in the Reynoldsville district: or converting this into a royalty basis on percentage of yield, about double, the Reynoldsville coal being worth about 4 cents per ton in the ground as against 8 cents per ton for Connellsville.

There will be differences also in mining-costs outside the differences in the mining-scale. All the coal to be won at the Reynoldsville plant will be by shafting 150 ft.; and all the drainage of the mines must be pumped that height to the surface.

On the other hand the Connellsville plant, though a slope with an ultimate depth of 200 ft., will be largely self-draining, due to the coal-seam outcropping for some distance in an adjoining valley; and while the balance of the territory must be

worked by shafts of from 225 to 250 ft. in depth, it can hardly be said that with a given equal output in the two regions there will be any appreciable difference in the item of pumping or hoisting at either plant.

Both regions enjoy extremely favorable structural features, thereby minimizing the costs of underground haulage and movement of coal to the surface. The minor differences in this respect between the two regions cannot be expressed in figures, although the Connellsville district is apt to prove the more regular and uniform in structure of the two and, therefore, the cheapest to work and to maintain.

The main differences in cost of mining coal will result, of course, from inherent differences in the texture of the coal and the ease with which it yields to mining, and from the different rates paid for digging and labor in the two fields.

In the manufacture of coke, in the same sized oven and in the same number of ovens, differences will also manifest themselves: 1. In the price of raw coal charged to the oven, and, 2. In the yield of coke per ton of coal.

In these two particulars the Connellsville district has a decided advantage, amounting to about 30 cents per ton in the cost of coal (after deducting proportional royalties in each case) and nearly 10 per cent. in the yield of coke.

The former is necessarily a variable item, for the rates for mining and labor in connection with mining, while always different in the two regions, do not vary in the same proportion, nor at the same time; the laws of supply and demand affect this item. But the ratio of yield remains fixed, due to inherent chemical differences in the two coals, and may be safely estimated to approximate 10 per cent. in favor of Connellsville.

Again, methods of mining may affect the net-cost of coal per ton for the reason that considerable differences may result from the partial or total adoption of machine-mining in one or the other district, and in some measure eliminate the natural advantages of the Connellsville district through its possessing the thicker, more regular and more persistent coal-seam.

But assuming identical methods of mining in each field, there is always a large difference in cost of coal charged to the ovens in favor of Connellsville, which is further emphasized by the difference of yield in coke.

For convenience in reporting the coke-product, Pennsylvania is divided into several districts,[6] among which the principal ones are:

1. The Connellsville district, embracing the ovens of Westmoreland and Fayette counties, from just south of Latrobe to Fairchance in the Connellsville basin.

2. The Lower Connellsville region, entirely in Fayette county, an extension southwest of the Connellsville basin proper, but embracing territory to the west of the Fayette anticlinal axis and in the adjoining Lisbon coal-basin, from Perryopolis on the north, including the Redstone and Klondike fields.

3. The Upper Connellsville district, sometimes called the Latrobe district, extending northeast from the Pennsylvania railroad through the Connellsville basin.

4. The Reynoldsville-Walston district of Jefferson county, including the ovens near those towns and some few ovens in Clarion county.

5. The Allegheny Mountain district, including the ovens in the vicinity of Johnstown and those lying along the Pennsylvania railroad east of Blairsville and some few plants in Somerset county.

The official returns of production, average value and yield of coal in coke in these several main districts are given[7] for the period of 1898 to 1902, inclusive, in Table II.:

To show the values of coke per ton more concisely, on the basis of figures compiled by the Government, Table III. has been prepared.

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[6] "These divisions are based to some extent on geographic boundaries, but also upon the quality of the coal mined and the coke produced." E. W. Parker, *Report on the Manufacture of Coke*, U. S. Geological Survey, 1902.

There are several additional districts—Greensburg, Irwin, Broadtop, Pittsburg, etc.—whose total production of coal only amounts to about 10 per cent. of the output used in coking.

[7] *Mineral Resources*, U. S. Geol. Survey, 1901 and 1902. (In the early part of January there were no official returns as yet made for the past year, 1903, and even in previous years the average figures covering costs of mining and coking in these districts have apparently been deduced from grouping together the returns from a large number of plants—good, bad and indifferent—and bearing no relation, physically or structurally, to the two particular plants under consideration.)

TABLE II.—*Production, Average Value and Yield of Coke in Various Districts in Pennsylvania.*

District.		Coke Produced.	Value Coke Per Ton at Ovens.	Yield of Coal in Coke.
		Short Tons.		Per Cent.
Connellsville	1898	8,315,350	$1.518	66.8
	1899	10,390,335	1.64	69.4
	1900	10,020,907	2.23	67.0
	1901	10,235,943	1.873	67.0
	1902	10,418,366	2.283	67.05
Lower Connellsville (a)	1900	385,909	2.05	66.5
	1901	1,116,379	1.784	66.9
	1902	1,899,111	2.475	67.2
Upper Connellsville	1898	403,045	1.34	63.0
	1899	609,893	1.62	65.3
	1900	690,449	1.996	66.2
	1901	569,511	1.815	66.8
	1902	936,854	2.34	66.3
Reynoldsville	1898	600,084	1.41	58.7
	1899	972,933	1.84	61.5
	1900	625,553	2.15	56.0
	1901	589,577	1.988	55.7
	1902	689,890	2.06	55.1
Allegheny Mountain	1898	378,410	1.35	66.0
	1899	478,340	2.01	65.5
	1900	557,184	2.26	63.6
	1901	548,076	2.03	63.4
	1902	644,053	2.76	66.7

(a) Only reported since its existence from 1900.

TABLE III.—*Estimated Value of Coke at Ovens in the Five Principal Districts.*

	1898.	1899.	1900.	1901.	1902.	Average Value.
Connellsville	$1.518	$1.64	$2.23	$1.873	$2.283	$1.91
Lower Connellsville (three years)	.....	.....	2.05	1.784	2.475	2.10
Upper Connellsville	1.34	1.62	1.996	1.815	2.340	1.82
Reynoldsville	1.41	1.84	2.15	1.988	2.06	1.89
Allegheny Mountain	1.35	2.01	2.26	2.03	2.76	2.08

While it is difficult to dispute the official figures given in Tables II. and III., which are presumably based upon the actual returns of operators, I question their entire accuracy as representing actual commercial results and returns.

For instance, it is generally admitted—all things being equal—that there is a regional difference of 50 cents per ton

between Allegheny Mountain coke and Connellsville coke in favor of the latter, and similarly a difference of not less than 10 cents between the price of the typical Connellsville product and the coke produced in either the Upper or the Lower Connellsville fields.

And while such figures may truly represent the general average results of all plants, the average prices as quoted cannot truly represent the actual coke-values or the normal commercial returns of certain mines.

From other figures given in the Government reports the average price of Connellsville furnace-coke[8] for the same period, based on contract-sales, is given:

TABLE. IV.—*Price of Connellsville Furnace-Coke Based on Contract-Sales.*

	1898.	1899.	1900.	1901.	1902.	Average Price.
Connellsville furnace-coke.....	$1.675	$2.185	$2.89	$1.99	$3.17	$2.382

To arrive at an independent estimate of costs and prices in the Reynoldsville and Lower Connellsville fields from which to figure the normal returns from the two 500-coke-oven plants, it would seem best to take (1) the relative mining-rates prevailing in the two districts and calculate cost of coal to the oven, and (2) from the costs of the raw coal to oven, calculate the net-cost of coke on the basis of yield.

The relative rates paid during January, 1904, for mining, calculated total cost of coal, royalty or value of coal per acre, and total cost of coal to oven in the two districts, may be represented as shown in Table V.

TABLE V.—*Relative Rate Paid Per Gross Ton for Mining and Royalty, and Total Cost to Oven.*

	Mining-Rate, Gross Ton.	Cost of Coal, Gross Ton.	Royalty.	Total Cost to Oven Per Gross Ton.
Reynoldsville.....	66c.	86c.	4c.	90c.
Connellsville.....	34c.	52c.	8c.	60c.

[8] Foundry-coke usually brings from 25 to 50 cents per ton more; but occasionally the difference is from $1 to $1.50 per ton.

Connellsville pays by the 100-bushels measure. It is a notorious fact that generally no account is taken of the weight of coal charged to an oven. The miner is paid by measure; but though paid for a 40-bu. wagon he frequently loads from 50 to 60 bushels. The operator, of course, gets the benefit of it; so does his cost sheet; but the scientific value of his work is lost and false deductions follow as the net result of unstable and unknown first cost of raw material. Roughly calculated, 100 bushels = 4 tons, for which to-day $1.10 is paid for mining room-coal or 27.5c. per ton. The correct cost, however, would be more nearly figured in this way: 100 bushels = 7,600 net pounds = 3.8 net tons or 3.39 gross tons, for which $1.10 is paid, or about 32c. per gross ton for room- and rib-coal.

The usual system, however, is to calculate 4 tons to the 100 bushels. The scale of wages paid from 1899–1903, inclusive, with its varying prices for room-, heading- and wet-heading coal, complicates the problem of averaging the cost of all coal. As there is about 55 per cent. of room-coal and 45 per cent. of heading- or entry-coal mined, in Table VI. I have given under the respective headings: first the mining-rate per 100 bu.; next the converted price per gross ton as above calculated, and, finally, the average cost per gross ton on relative percentages of mining-rate in each year.

TABLE VI.—*Scale of Wages Paid for Mining.*

Period.	Year.	Room-Coal.		Heading-Coal.		Wet-Heading Coal.		Average Cost Gross Ton.
		100 Bu.	Gross Ton.	100 Bu.	Gross Ton.	100 Bu.	Gross Ton.	
Jan. 1 to Apr. 30...	1899	$1.05	30.5c.	$1.20	34.8c.	$1.25	36.3c.	34.43c.
Apr. 30 to Dec. 31.	1899	1.125	32.7c.	1.275	37.1c.	1.325	38.5c.	
Jan. 1 to Feb. 28...	1900	1.125	32.7c.	1.275	37.1c.	1.325	38.5c.	37.99c.
Feb. 28 to Dec. 31.	1900	1.25	36.4c.	1.40	40.8c.	1.45	42.2c.	
Year.....	1901	1.25	36.4c.	1.40	40.8c.	1.45	42.2c.	38.69c.
Year.....	1902	1.25	36.4c.	1.40	40.8c.	1.45	42.2c.	38.69c.
Jan. 1 to Dec. 16...	1903	1.35	39.3c.	1.50	43.6c.	1.55	45.1c.	41.26c.
Dec. 16 to Dec. 31..	1903	1.10	32.0c.	1.25	36.3c.	1.30	37.8c.	

But in addition to the variation in wages paid for mining, all other labor employed—both in producing coal and manufacturing coke—varied also, and not proportionately; the actual total cost of producing coal is therefore somewhat difficult to

compute. The calculation for coke costs is even worse, for in this region the coal charged to oven is never weighed, the standard of measure being the wagon of 40 bu., which may mean 50 or even 60 bu. calculated from the weight.

Using the figures given in Table V. as a basis for computing coke cost, and also remembering that the yield of coke in the raw coal is about 57 per cent. in the Jefferson county field as against 67 per cent. in the Connellsville field, the total cost of coke in each district may be approximately estimated as follows:

Reynoldsville:	Coal 1.7 tons @ 81c. net ton = . .	$1.38	
	Charging, leveling, drawing and labor,	0.40	
	Salaries, supplies and depreciation, .	0.05	$1.78
Connellsville:	Coal 1.5 tons @ 54c. net ton = . .	$0.81	
	Charging, leveling, drawing and labor,	0.32	
	Salaries, supplies and depreciation, .	0.05	1.18
Difference of cost in favor of Connellsville, . . .			$0.60

In the Walston-Reynoldsville district, the Clearfield mining-rate applies, from which the estimated total cost of coal, including royalty of 4c. per ton, may be calculated as shown in Table VII.

TABLE VII.—*Cost of Mining Per Ton in the Walston-Reynoldsville District.*

Year.	Mining-Rate.	Cost Per Gross Ton.	Cost Per Net Ton.
1899	50c.	68c.	61c.
1900	50c.	68c.	61c.
1901	60c.	82c.	74c.
1902	60c.	82c.	74c.
1903	66c.	90c.	81c.

From the figures in Table VII. the cost of the coke per net ton may be calculated as in Table VIII., on basis of 57 per cent. yield.

TABLE VIII.—*Calculated Cost of Coke in the Walston-Reynoldsville District.*

	1899.	1900.	1901.	1902.	1903.	Average.
Cost of coke at ovens.....	$1.35	$1.35	$1.60	$1.60	$1.75	$1.53
Selling price at ovens.....	1.85	2.15	2.00	2.05	2.25	2.06

Similarly, the estimated figures for coal and coke in the Connellsville region, with 8c. per ton royalty added, are given in Table IX.

TABLE IX.—*Cost of Mining Per Ton in the Connellsville District.*

Year.	Mining-Rate.	Cost Per Gross Ton.	Cost Per Net Ton.
1899	34.5c.	57c.	51.0c.
1900	38.0c.	62c.	55.8c.
1901	38.7c.	63c.	56.7c.
1902	38.7c.	63c.	56.7c.
1903	41.3c.	70c.	63.0c.

From the figures in Table IX. the cost of the coke per net ton may be calculated as in Table X., on basis of 67 per cent. yield.

TABLE X.—*Calculated Cost of Coke in the Connellsville District.*

	1899.	1900.	1901.	1902.	1903.	Average.
Cost of coke at ovens.....	$1.10	$1.18	$1.20	$1.25	$1.35	$1.216
Selling price at ovens.....	1.82	2.46	1.90	2.34	3.00	2.304

A summary of these estimates for the five years, 1899 to 1903, is as follows:

Average price Connellsville coke, ...	$2.304	
Average cost Connellsville coke, ...	1.216	
Average profit per ton, ...		$1.09
Average price Reynoldsville-Walston coke, . .	$2.06	
Average cost Reynoldsville-Walston coke, . .	1.53	
Average profit per ton, ...		0.53
Average difference in value per ton in favor of Connellsville,		$0.56

The only additional factors accessible, bearing upon the question of costs and values, are contained in a review of the Connellsville Coke-Production for 1903.[9]

Table XI., issued by the same authority,[9] shows statistically

[9] *Connellsville Courier*, Jan. 14, 1904.

the growth and value of the coke industry during the past decade.

TABLE XI.—*Connellsville Coke Production:* 1894–1903.*

Year.	Total No. of Ovens.	Tons Shipped.	Average Price.	Gross Revenue.
1894	17,834	5,454,451	$1.00	$5,454,451
1895	17,947	8,244,438	1.23	10,140,658
1896	18,351	5,411,602	1.90	10,282,043
1897	18,628	6,915,052	1.65	11,409,835
1898	18,643	8,460,112	1.55	13,113,179
1899	19,689	10,129,764	2.00	20,259,528
1900	20,954	10,166,234	2.70	27,448,832
1901	21,575	12,609,949	1.95	24,589,400
1902	26,329	14,138,740	2.37	33,508,714
1903	28,092	13,345,230	3.00	40,035,690

* The Allegheny Mountain district furnishes an additional check on these figures. From data furnished by the largest operator in that field—where the Clearfield rate for mining prevails as in the Reynoldsville district—he estimates that at the present rate of mining (66c. per gross ton) 25c. additional should cover all costs, other than mining labor, without royalty. Under such circumstances, Reynoldsville coal, charged to ovens, might be figured at 85.5c. per net ton, though I have used 81c.

With the cheaper rates of mining which prevailed in 1899, 1900, 1901 and 1902, perhaps 20c. and 22.5c. might cover outside costs, haulage, superintendence, repairs, supplies, etc.

This operator's estimates for making coke in this district are:

1899: Av. cost of coke, $1.35 per net ton	Sales 6 mos., $1.40; 6 mos., $1.60–$2.00
1900: Av. cost of coke, $1.50 per net ton	Sales during year, . @ $2.25 to $3.00
1901: Av. cost of coke, $1.65 per net ton	Sales during year, . @ $1.80 to $2.50
1902: Av. cost of coke, $1.90 per net ton	Sales 6 mos., $2.50; 6 mos., $4.00–$10.00

The average cost for the four years quoted, 1899–1902, would therefore be $1.60 per ton as compared with $1.53 for the Reynoldsville average for five years; and there are sufficient economies in mining in the Jefferson county field to account for a difference of 10 cents per ton of coke.

The figures in Table XI. ought to be authoritative, for the reason that they are carefully compiled each year and are generally accepted as a correct exhibit of the region's industrial returns, and yet the figures of production (shipments) and the estimate of the average price of coke differ materially from those published by the U. S. Geological Survey as shown in Table XII.

TABLE XII.—*Difference in the Reports of the Courier and the U. S. Geological Survey.*

Connellsville Region. Lower Connellsville.		Coke Produced. Short Tons.	Total Value at Ovens.	Av. Price Per Ton.	Av. Both Estimates.*
1898.	Courier†.....	8,460,112	$13,113,179	$1.55	$1.55
	U. S. Geol. Survey....	8,315,350	12,626,292	1.52	
1899.	Courier.....	10,129,764	20,259,528	2.00	1.82
	U. S. Geol. Survey....	10,390,335	17,075,411	1.64	
1900.	Courier.....	10,166,234	27,448,832	2.70	2.46
	U. S. Geol. Survey‡...	10,406,816	23,176,318	2.22	
1901.	Courier.....	12,609,949	24,589,400	1.95	1.90
	U. S. Geol. Survey....	11,352,322	21,164,396	1.86	
1902.	Courier.....	14,138,740	33,508,714	2.37	2.34
	U. S. Geol. Survey....	12,317,477	28,486,501	2.31	
1903.	Courier.....	13,345,230	40,035,690	3.00	3.00

* This column of combined average selling prices, from the two best authorities, has been used in Table X., rather than the figures of contract-sales in Table IV., as being more normal and more conservative, as well as more representative, of the whole district.

† The *Courier* returns are presumably based on short tons, as that is the standard for all sales of coke. There also should be no appreciable difference between production and shipments, though there may be some slight consumption in the region, not billed on cars.

‡ The average price for this and succeeding years (Lower Connellsville field not being previously reported) is figured from the average prices given by the Survey (see Table II.) for each of these districts, upon proportion of relative production.

On the question of average price of coke for 1903 the *Courier* says:

"It is a very difficult matter to estimate the average price of Connellsville coke for the year just ended, because of the wide range of prices during the year and the unusual number of operators in the market. Heretofore, the bulk of the outside coke was handled by the H. C. Frick Coke Co., and there were really but few sellers in the market. This concern held the prices steady, and market quotations mean just what they said. The Frick company retired from the market at the close of 1902, and this has been a free-for-all year for the other operators. The market quotations after the first three months were persistently shaded and, finally, recklessly cut until they were reduced from $5.50 @ $6 in the beginning of the year to $1.50 @ $1.75 at its close. This result awoke the operators to the importance of getting together for the purpose of protecting prices, and a movement of this kind is now well under way and will doubtless be successful. It is now conceded that Connellsville coke ought never to sell under $2 per ton."

Summary.

From the foregoing statements, carefully verified by inquiry through every avenue of information open to me, and checked as far as possible by my own data and estimates, the following deductions have been drawn:

Given two coke-plants of equal capacity, 500 bee-hive coke-ovens, 12 ft. in diameter, with equal facilities for mining and transporting coal, and each plant converting its entire product into coke, the operations for the past five years, 1899–1903, working an average of about 250 days each year, would have yielded, approximately, the following results:

1899–1903: *Connellsville Region Plant:* 500 *Ovens.*

Total coal required for coking, 1899–1903,	2,238,806 short tons
Total coke produced @ 300,000 tons per year,	1,500,000 short tons
Average yield of coal in coke,	67 per cent.
Average price of coke during five years,	$2.304
Average cost of coke during five years,	1.216
Net profits on total product @ *average* of $1.09 per ton,	$1,635,000

1899–1903: *Reynoldsville Region Plant:* 500 *Ovens.*

Total coal required for coking, 1899–1903,	2,631,579 short tons
Total coke produced @ 300,000 tons per year,	1,500,000 short tons
Average yield of coal in coke,	57 per cent.
Average price of coke during five years,	$2.06
Average cost of coke during five years,	1.53
Net profits on total product @ *average* of 53c. per ton,	$795,000

In other words, on the same amount of coke product during the past five years, the Connellsville plant should have cleared $840,000 more than the Reynoldsville plant, receiving an average profit of $1.09 per ton as against 53c., or more than 50 per cent. greater income, and with a saving of nearly 400,000 tons of coal, owing to the greater percentage of yield in coke.

Manifestly the Connellsville plant has still further economies attached to its operation in maintenance, management and greater accessibility to market, though greater perhaps than the real fuel efficiency of the two cokes would show in their respective consumption.

A discussion of the relative value of the two cokes to the consumer would be a natural and valuable complement to the present paper.

Concrete in Mining and Metallurgical Engineering.

BY HENRY W. EDWARDS, GRAND JUNCTION, COLO.

(Atlantic City Meeting, February, 1904.)

CONCRETE is not a new, nor even a modern substance. Important structures built by the old Romans before the commencement of the Christian Era are to-day sound and solid—for example, the dome of the Pantheon in Rome, 142 ft. in diameter. I do not propose, therefore, to discuss the durability of the material, as in the mining and metallurgical business, structures, as a rule, are not calculated to endure for two thousand years. I do wish to urge, however, that much of the stone masonry around mines and smelters would have given better results if concrete had been used in place of stone. Concrete looks better; is stronger, cheaper, more durable, more quickly built, and above all requires no skilled labor, except that of the carpenter, who, like the poor, is always with us.

Well proportioned concrete, that is to say, having well proportioned ingredients, may be safely relied upon to withstand a compressive or crushing strength of 50 tons per square foot. In most American cities, however, the building laws limit it to the absurdly low figure of 16 tons. This restriction is probably due to the political influence of the building-trades unions, which object strongly to its too free use, undoubtedly owing to its not requiring the services of skilled workmen. I hope to show that quite important construction work may be undertaken without the aid of the gentleman with the trowel. In some of the isolated districts of the West and Southwest this is no mean advantage.

The weight of good concrete, dry, is from 130 to 160 lb. per cubic foot.

INGREDIENTS OF CONCRETE.

Concrete is composed of four ingredients; *viz.*, 1. *Crushed Stone;* 2. *Sand;* 3. *Cement;* and 4. *Water.*

1. *Crushed Stone.*—The stone is usually crushed to a size that will pass through a ring of from 2 to 2.5 in. in diameter. It is not necessary or even beneficial to screen the material. Soft rocks, as sandstone or soft porphyry, are not to be recommended unless the concrete is to bear either no load, or an insignificant one. Slag roughly broken to from 2- to 2.5-in. size is in every way suitable. The only objectionable ingredient in slag is calcium sulphide.

2. *Coarse, Gravelly Sand.*—Jig-tailings, both coarse and fine (7 mm. and under), give good results, especially the 2.5-mm. size. A mixture of all sizes with the from 1.5- to 2.5-mm. size predominating is perhaps the best form in which to use jig-tailings. Vanner-tailings and stamp-battery tailings are not so advantageous, being usually too fine in size. Granulated slag is excellent.

3. *Cement.*—Cement is essentially a combination of lime with aluminum silicate. A few deposits of rock are known in which limestone and clay occur in about the proper proportions to form cement, and in these cases it is only necessary to calcine and pulverize such material to obtain the so-called "Natural Cement." Louisville, Milwaukee and Akron are well-known points supplying such a product. The calcination has to be very skilfully conducted at a low heat. If burnt at too high a temperature, it fuses to a slag which when ground exhibits no cement qualities. Moreover, the rock but seldom contains the true proportion of lime and alumina, and contains, as well, other substances of an undesirable nature, as, for instance, magnesia. For these reasons the natural cement, while much cheaper in first cost, is usually much less effective than the artificial or Portland cement. If any doubt exists as to whether any particular sample of cement is natural or artificial, a test for magnesia will usually settle the matter. In Portland cement, magnesia is only a slight accidental impurity (2 per cent. or so), while in natural cement it sometimes exceeds 10 per cent.

The artificial or Portland cement is usually manufactured under the supervision of a chemist, the ingredients being adjusted minutely to a definite standard. It is, therefore, much

more regular and reliable than the natural product, and in making concrete, two of Portland equals three of natural cement. This ratio should be borne in mind in ascertaining the values of different cements laid down at the work. In the testing of cement for probable value, the determination of its fineness should not be omitted. All material exceeding 100-mesh size is likely to have no value as cement, and the portion which does not pass a sieve of this fineness is so much waste material. Wire-cloth of this fineness is so easily distorted when mounting on the sieve (independently of what it may have gone through before it reaches the laboratory) that it is seldom really 100 by 100 meshes to the inch, and is often very wide of the mark. This fact should be duly noted by all.

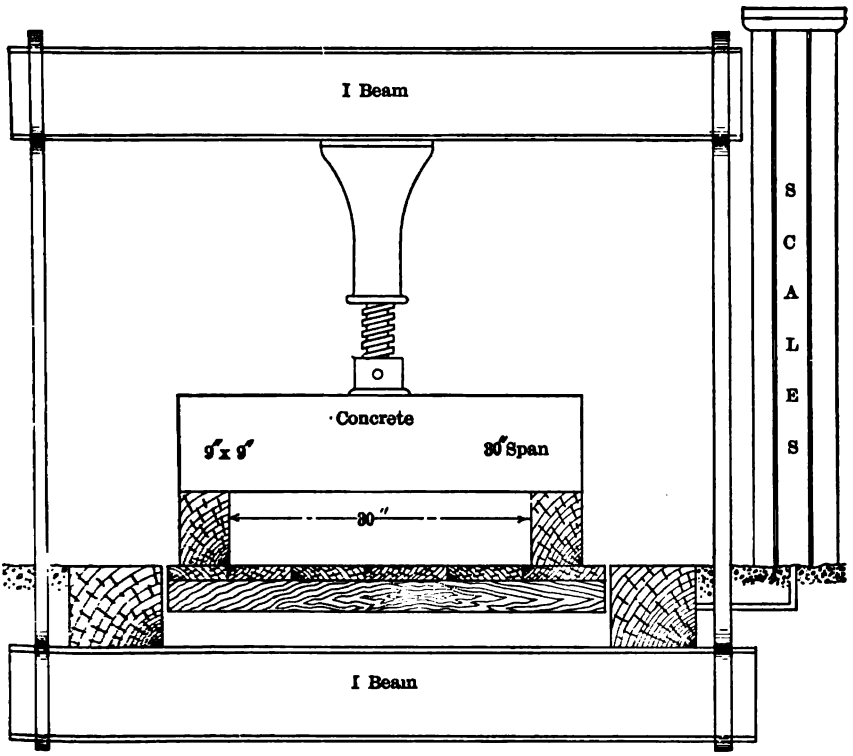
Portland cement only is referred to in the following mixtures. There is a considerable difference between the various brands of Portland cement, and these differences are accentuated by age and care in storage. Before finally deciding on any large purchase it is well to experiment with the cement commercially possible in the locality.

Concerning the means for testing concrete, apparatus for measuring the ultimate crushing strength of a sample block of concrete is hardly to be found in any mining or metallurgical equipment; but a comparative test can easily be made of its breaking strength. As a substitute for such a testing-machine nothing is better than the ordinary 5-ton platform scales with a good strong screw-jack. Fig. 1. shows the method of testing I usually follow, and the details of the work are sufficiently obvious without a detailed description—the pressure exerted by the screw-jack being read off on the scale beam at the moment the cracks appear on either of the visible sides of the test piece.

Several preliminary series of mixtures of the various ingredients in various proportions should be made—a series with each brand of cement. The mixtures should be rammed in a wooden box, 9 in. square and a little more than 30 in. long, inside measurement, and left for several days or weeks until set. Several batches should be mixed, varying with the nature and the proportions of the ingredients obtainable, and several casts made of each batch. A series of mixtures is suggested as follows:

Cement.	Sand, Fine Gravel or Tailings.	Crushed, Unscreened Rock.
Parts.	Parts.	Parts.
1	0.5	7
1	1	6
1	2	5
1	3	4
1	4	3

FIG. 1.



METHOD OF TESTING CONCRETE-BLOCKS WITH A SCREW-JACK AND PLATFORM-SCALES.

For the best and strongest work, it is usual to have the cement about 15 per cent. of the total quantity of the ingredients. Generally, however, there should be three different classes of concrete, according to the use for which the work is designed.

1. *Strong:* Containing about 15 per cent. of cement, for retaining walls, flues, culverts, arches and walls for dust chambers, arch-work in general and foundations in wet places.

2. *Medium:* Containing about 10 per cent. of cement, for engine and machinery foundations on good dry ground, floors, bottoms of flues and dust chambers, stack and furnace foundations.

3. *Poor:* Containing about 7 or 8 per cent. of cement, for leveling the bottoms of excavations previous to commencing foundations proper and for all foundations and underground work where the weight to be supported is not to exceed 6 tons per sq. ft.

Both medium and poor grades may be diluted still further, without diminishing the ultimate strength, by embedding large boulders provided the concrete be well rammed around them.

For testing purposes it is only necessary to work upon one of the above-named varieties, usually the strong. At the end of a week, one or two test-blocks from each series may be tested, leaving others to be tested at the end of four weeks.

In an article on the subject of concrete,[1] Mr. C. R. Neher says, in speaking of concrete made from a mixture of copper slag and gravel:

"My first tests of the slag-mixture gave only about 80 tons per square foot in seven days, and appeared to demonstrate a low value for the slag, but examination of the fracture showed an excess of gravel and a fracture through the spaces where the most gravel existed. A slight diminution in the quantity of gravel gave results of over 140 tons in seven days."

Neher refers to compression tests on a 12-in. cube.

This statement is quoted in order to show that I am not alone in urging the importance of testing various proportions of the ingredients at hand before beginning the work. This is one of the cases in which haste must be made slowly.

The search for ingredients for concrete is generally limited to material found in the immediate vicinity of the work. It would not be of much use to go into the question of absolute figures on typical crushing weights; and as the figures sought for are comparative only, the experimenter is confined to seeking the best material attainable in his vicinity. Any block which will stand up to a 2-ton breaking-test, as above described, can be pronounced very good, although a 4-ton test after drying for several months is not too much to expect. Hence, the whole question is reduced to, "How little cement can be used to obtain the desired strength?" It is to be noted that the cement is the most costly ingredient of the concrete mixture.

[1] *Architects and Builders' Magazine*, August, 1901.

Concerning the total quantity of material required to fill a given space, this case is one in which two and two do not make four. For example, two parts of sand and four parts of crushed rock, mixed together do not give six volumes of mixture, for the reason that a portion of the sand will fill up the spaces between the larger pieces of rocks. Moreover, on adding one part of cement, it passes entirely into the voids between the sand and rock particles, and adds nothing to the total bulk. Also on wetting and throughly ramming the concrete in place, it will pack to a bulk of four parts,—or, approximately, equal to the original bulk of the crushed rock. Therefore, in estimating the quantity of ingredients required for a piece of work, it is necessary to provide a cubic foot of crushed rock for each cubic foot of completed structure, counting that the sand and cement do not in any way contribute to the final bulk.

Water is dealt with in the section devoted to "mixing," which is given later in this paper.

The use of lime as well as cement is the subject of much difference of opinion. My experience is that it is quite neutral, appearing merely to delay the setting of the concrete.

As a general rule it may be stated that the effect of an excess of fine material is weakening, but it gives a good, smooth, finished appearance to the work.

Before leaving the subject of ingredients, I ought to point out a curious property of Portland cement, which is of importance, especially if the work is to be let out on contract, the contractor providing the materials. When a package of cement is opened and emptied, it can be dusted out so that the bulk is increased quite one-third. I will defy anyone having once emptied a barrel of cement to replace the whole of it in the same barrel without considerable effort.

At first, until confidence is gained with unaccustomed material, the best way to measure the sand and the rock is to use one or more empty cement barrels, having the heads and bottoms knocked out. When placed upon the ground and filled the barrel can be lifted off and the operation repeated until the desired volumes of sand and rock have been measured, then a barrel of cement in the original package can be added to the mixture. For this procedure the cement should not be bought in sacks for the reason that it would be almost impossible to

empty the sacks into a barrel without largely increasing the bulk, unless, of course, the exact number of sacks per barrel had been accurately ascertained beforehand.

During the whole process of making concrete, it is essential to avoid admixture with earth, clay, dirt, sawdust, chips or other undesirable material.

CONCRETE CONSTRUCTION.

Mixing.—Unquestionably, the mixing of the ingredients by machine is much better than by hand, and in my experience if more than 80 cu. yd. of work be required, it will pay to install a mixing apparatus. Several manufacturers supply very complete mixing outfits, but my experience has been confined to home-made mixing-machines. The apparatus I have used, illustrated in Fig. 2, is simply a cubical wooden box lined with No. 10 sheet-steel and having an iron manhole at one corner. The box is mounted on two corners or trunnions, one a piece of 3-in. pipe through which the water is introduced, and the other connected by means of a gear-wheel and pinion to a hand-crank. The manhole is turned up to receive the charge from the hopper and is then fastened down. The box is then revolved a few times to give the ingredients a dry mix. The necessary quantity of water is then introduced by hose and nozzle through the hollow trunnion and the revolving continued as long as required. The manhole is removed and the box partly rotated, so that the contents are discharged into a wheelbarrow for transportation to the work. Too long a trip for a wheelbarrow is to be avoided, unless precautions are taken, because the liquid separates from the mass, and, if in a leaky wheelbarrow, is lost, or if in a tight iron wheelbarrow, the workman often empties the "slush," which is mostly cement, elsewhere than where it ought to go. In order to avoid this action, it is best to dump the material on to a small platform of sheet-iron, remix it by turning over, and then shovel it to its proper place.

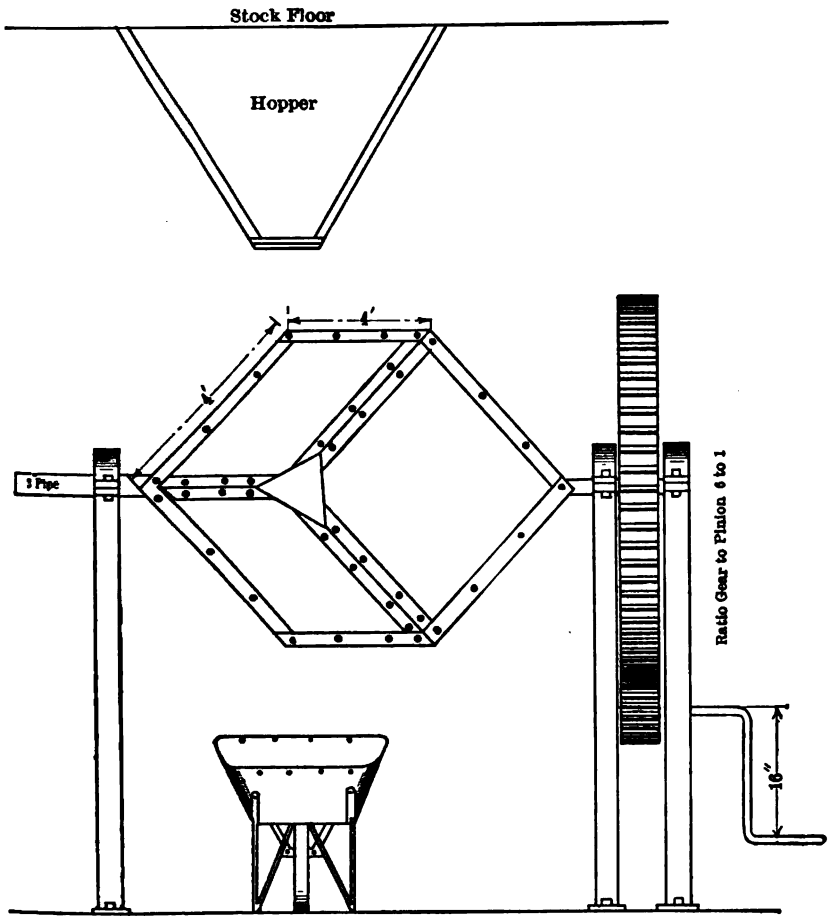
Machines on the pug-mill type, which deliver the concrete after a definite and invariable number of revolutions, are not so satisfactory as those of the box-type, which permit the mixing to be continued as long as desired.

Mixing by hand consists merely in turning a shallow bed of the ingredients over and over with shovels on a platform. The shovel should go fully down to the platform and the material

be turned completely over. A sheet-iron platform lightens the labor.

The ideal mixing is when each separate particle of rock and sand is coated entirely with cement, and the nearer this ideal is reached the better will be the concrete.

FIG. 2.



A HOME-MADE CEMENT-MIXER.

It is also essential to have all parts of the batch equally wet, otherwise by having some parts wet and some parts dry, the drier part will set before the wetter and thus cause stresses within the mass, which may sooner or later lead to ugly cracks.

With reference to the quantity of water required, this depends largely upon the climate and the season. In a hot, dry

climate, Arizona for instance, much more water is necessary than in a moist, cool climate; also in summer more water is required than in winter. I have experimentally made the mixture so wet that on stepping on the fresh concrete, one would sink in over the shoe tops, yet, on drying, the wet concrete had a smoother appearance than similar concrete containing less water which required to be heavily tamped; furthermore, I have never seen any evidence which led me to doubt the strength of the wet concrete. On the other hand in cool, moist weather, I have used the concrete so that it appeared only slightly moist, and persistent beating was required to force the water to the surface.

In general and within reasonable limits, the wetter the mixture the easier is the work of thorough mixing. It is best, perhaps, to follow a medium line, but no fixed rule can be laid down, for the reason that different kinds of rock and cement require different quantities of water. It is easier to make the mistake of having it too dry than too wet. A good rule to follow is to have the concrete so wet that it will shake like a jelly when being rammed into place with a heavy beater, and will only just support the beating-tool. In dry weather only should it be fully up to this limit; it may be kept somewhat stiffer in wet, cool weather. If spread in shallow layers (floors, for example,) the mixture should be much wetter then when laid in a mass, otherwise it will dry up before chemical action begins.

In retaining-walls I prefer to use the mixture as dry as I reasonably can, because an excess of water, on evaporation, leaves the concrete somewhat porous, and thus allows seepage-water to pass through it.

If concrete impervious to water be required (tanks, reservoirs, etc.), it is necessary to use a somewhat richer mixture, *i. e.*, one containing more cement, and to have it thoroughly mixed and tamped. The ideal condition for this class of material is to proportion the coarse and fine ingredients so as to eliminate all the voids.

Laying and Beating the Concrete.—For foundations and walls the material should be laid in layers not exceeding 6 in. in depth; if thicker than this, it is apt to lose a part of the benefit of the labor of beating—the effect of the blows not penetrating to the bottom of the layer. A convenient beater is made of an iron-casting, 6 in. square, having a handle of 1-in. round

iron, or 1.5-in. pipe, about 5 ft. in length, which should weigh in all from 20 to 25 pounds.

In order to avoid horizontal joints, it is very necessary that each layer should be put in before the previous one has set, and that not more than half an hour should elapse between completing, mixing and depositing. If for any reason, however, the work has to be interrupted, a number of large rocks or old rail-ends set vertically should be partly embedded in the last layer so that the projecting parts will provide a grip for the succeeding layers. Even this, however, should not be permitted near the top of the work. On continuing the work, the layer that has set should be thoroughly swept and wetted, first with water and then with thin cement-grout. It is of great advantage to have the work continue day and night until completed—the men working in squads.

In a long and heavy retaining-wall, it is best to divide the length into sections, because vertical joints do not affect the strength to the same extent as do horizontal ones. In any event, it is best to divide a large wall into panels, otherwise, owing to the expansion and contraction of concrete under changes of temperature (the co-efficient of expansion of concrete is but little less than that of iron), cracks are apt to develop, and thus spoil the appearance of the work. This division of the work into panels causes the cracking to take place along the vertical joints where their appearance is not so evident or disfiguring. A panel should be of a size that can be completed in a day's work.

During the progress of the work it is a good plan to embed in the concrete old wire-rope, chains, or rails, which help to bind the concrete together and strengthen it. Concrete adheres very tenaciously to iron and seems also to preserve it from rust.

The Crib, or Form.—Unless the concrete is being filled into an excavation below the ground-level, it is necessary to construct a crib of timber, the interior of which has the shape of the exterior of the completed work. There is scope for much ingenuity in the arrangement of these cribs, not only to produce the necessary shape and size, but to have them so stiff that no amount of beating of the concrete will cause the boards to bulge or to shift. Also, for economy's sake, the work should be so designed that the same lumber may be used repeatedly.

I have found that tongue-and-groove boards are not as satisfactory as plain boards, or planks, with straight, planed edges. Cement works into the tongue and groove and then, when swelling occurs from the contact with the wet concrete, the lumber appears to warp irregularly, and to leave unsightly marks of each plank on the face of the finished work. Unless braces are put in frequently, I have not found any advantage in using 1-in. boards, and I much prefer 2-in. ones.

Rounded corners are formed by inserting strips of "cove" molding in the crib, and for corners of walls around which there is much traffic, it is well to place inside the crib a bent strip of stout, sheet-iron, provided with bolts. The concrete is beaten around and behind these bolts embedding them and the sheet-iron snugly in place.

Concrete should never be dropped from a height for the reason that, invariably, the mortar and stone separate, and weakness results, an effect which I have noticed to be more pronounced with broken slag. For example, if it be necessary to get to the bottom of a deep excavation, it pays to lower the concrete in a tub or bucket, or if the excavation be large, to lower the wheelbarrow itself. In a case of this kind, I find it pays best to discard the wheelbarrow entirely, mount the mixing-gear close at hand, erect a temporary derrick, which will command not only the mixing-gear, but the area of the whole excavation, and then carry on the work with a tub.

Strength.—Concrete that has been made with care is superior in strength to the best masonry, but any slovenliness or neglect in the mixing or wetting, or in permitting a separation of the ingredients after thorough mixing will bring its penalty. Poorly proportioned ingredients, particularly an excess of fine material, also lead to weakness. The manner in which the best results may be ascertained experimentally by simple tests is described earlier in this paper.

I have tried the following experiment: A square slab of concrete 10 ft. by 10 ft. and 4 in. thick, having embedded within it old 0.5-in. mesh wire-screen, after seven months' seasoning was supported by placing a block under each of the four corners. The slab withstood without apparent bending a distributed load of 10 tons, and afterward a load of 3,000 lb. concentrated in the center.

The following quotation on the subject of strength is of interest:[1]

" . . . . A ring of 6- by 6.5-in. angle iron 15 ft. in diameter was used as a horizontal circular tie, within which a flat dome of concrete was built. The concrete was 6 in. thick throughout, and was formed of four parts of stone, two of sand and one of Portland cement. The dome rose 6.44 in. at the center, and it stood covered with 2 in. of damp sand for 58 days after it had been completed. A bed of sand was then laid over it to support a circular platform 5 ft. in diameter. About 60,000 lb. of bricks were laid on the platform before any deflection could be detected, and 80,000 lb. were supported for six days without any signs of failure. It was intended to test the dome to destruction, but more than 80,000 lb. could not be applied without danger to the men conducting the experiment."

It is to be noted that the writer of this statement says nothing about having iron- or woven-wire embedded in the concrete of this dome.

Cost.—It is difficult to give any very definite figures in regard to this very important subject because the prices of suitable rock and the facilities for crushing it, as well as the cost of sand or tailings and cement, will vary widely in different localities. I submit, however, figures which may be useful as a guide. In modifying these figures to suit conditions in any given locality, it is important to bear in mind that the magnitude of the work has considerable effect on the ultimate cost. For example, a piece of work like the experimental dome just described consumes a relatively small quantity of concrete-material, while the cost of the wooden form must have been relatively large. Compare this with a heavy retaining-wall where a crib has to be built to form one face only. It is best, therefore, to limit this discussion to the question of the cost of labor only; the remaining figures of the cost of the material and the crib can then be adjusted to suit the local conditions.

Presuming that the materials are delivered in railroad-cars convenient to the work, I find that an average cost—covering many jobs at different periods and locations—of unloading cars, mixing, arranging platforms and gangways, placing and beating concrete in place, is approximately equivalent to one man's wages for one day per cubic yard of structure. This estimate is sufficiently liberal to cover incidentals, such as wear

[1] *Engineering Record*, vol. 37, p. 168.

and tear of tools, lights at night, etc. Beyond this there yet remains the cost of cribs, excavations, scaffolding, lumber, etc.

About the cheapest work recorded in my note-book is a retaining-wall 16 ft. high, 92 ft. long, 3 ft. thick at the base, tapering to 20 in. at the top. The ingredients of the concrete were jig-tailings and picking-belt rock available on the spot, and cement at $2.15 per barrel. The lumber cost was $14 per 1,000 ft. and wages were 15 cents per hour. The total cost of the wall was 22 cents per cubic foot. This cost included the straightening up of the work after it had been roughly finished, and the supervision during the entire construction. About two-thirds of the lumber was credited as it was fit for other rough work.

Another entry in my note-book is the following:—A large engine-foundation of hand-broken slag, granulated-slag and cement was constructed at a total cost, including crib, of $7.75 per cu. yd. in place. In this instance, I have specified, not per cubic yard of material, but per cubic yard of finished work. It is necessary to bear this distinction in mind because, as before explained, a cubic yard of tamped and finished work requires more than a cubic yard of material.

I find that there is but little difference in the cost of hand-mixed concrete and machine-mixed concrete. This statement, of course, is restricted to the comparatively small structures built in the ordinary run of mining- and smelting-operations. For the construction of heavy railroad-bridges or other massive structures involving thousands of cubic yards, I believe there is now but little hand-mixed concrete used. Preference should be given to machine-mixed concrete, as it always gives better results.

Seasoning.—The seasoning of concrete is worthy of important consideration. The question is, "How long a period should elapse after laying the concrete before it should be loaded?" The answer depends on two conditions:—First, the season of the year, and second, the use to which the work is to be put. In summer time or in a dry winterless climate, concrete rapidly reaches its full strength, and it is best to retard its too rapid hardening by covering it with a layer of sand or dirt, which is frequently moistened. Under these conditions the concrete may receive its load in a week or 10 days, or even

sooner, if it is entirely below ground where the mass has no opportunity to spread. In more northerly localities where the winters are cold, concrete, if built late in the autumn or in the winter, will not be really solid until warm weather has set in. The influence of temperature on the hardening and seasoning of concrete is very distinct and must be taken into account.

The second condition which has to be considered before applying the load is the purpose of the structure. If the load be applied gradually, as for example, in the case of a retaining wall to be gradually filled up from behind, loading may commence after a week or two of warm weather. It would be unwise, however, to set an engine to work on a green concrete foundation, although there would be but little harm in mounting it in its place after a week or two have elapsed.

Thin work, such as floors and flues, naturally solidifies much more rapidly than heavy masses. I have used a floor on the third day after its completion, merely taking the precaution to place a few old boards where there would be the most traffic. Except in winter the cribs can be removed on the third day.

In heavy masses, hardening and seasoning will go on for many months. Beyond delaying the setting until warmer weather, frost, even if the wet concrete itself be frozen, does not appear to be in the least injurious.

Re-inforcing Concrete with Iron.—The imbedding of iron in concrete is almost as old as concrete construction itself. According to F. Von Emperger:[3]

" . . . . The writer knows of a case where iron rods were found perfectly rust-free, having been embedded in concrete below water level for 400 years. . . . . For the protection of iron, concrete is better than paint."

In 1897 there was some discussion in the columns of the *Engineering Record* on the effect of limestone in concrete, and a very general impression is left by the correspondence that limestone is injurious to iron embedded in concrete.

For re-inforcing retaining-walls nothing can surpass old rails set vertically and horizontally so that if the wall goes at all it has to go bodily, an effect not to be apprehended. For

[3] *Transactions of the American Society of Civil Engineers*, vol. xxxi., p. 453.

light work, such as roofs or walls of dust-flues and chambers, discarded coarse-mesh woven-wire trommel-screens are very suitable. In buying new material for this purpose a very favorite article is "expanded metal."[4]

It is light and strong and comes in large-sized sheets. In using it, half of the thickness of concrete is tamped in place, and before it has set the expanded metal is laid on and the remainder of the concrete tamped over, into and around it.

For foundations in bad ground, the area of the concrete may be very largely extended without fear of deflection by placing on the bottom of the mass a double series of steel beams, placed horizontally at right angles. I once had occasion to erect on very unsatisfactory ground a brick stack estimated to weigh 100 tons. The foundations, therefore, were extended and laid in the manner described above, by two layers of 8-in. steel I-beams 16 ft. long. In all, 36 I-beams were used, giving a foundation area of 16 by 16 ft., upon which was rammed about 3 ft. of first class concrete. Including the weight of this bed of concrete, the total stress on the under surface of the foundation was about 1,000 lb. per square foot, a most moderate pressure. This construction gave as firm a footing as could be wished.

Foundations Under Water.—Concrete may be deposited in still water and will harden there almost as soon as in the air. Evidently, however, it will not do merely to dump the concrete into a deep pool and trust in Providence for the rest. I have had very good satisfaction in the use of a home-made iron-funnel with a stem long enough to reach to the bottom, and which for convenience was cut shorter from time to time as the work progressed. The method of operating is to rest the outlet of the funnel on the bottom and to fill the stem and hopper with concrete, then by lifting it a little and moving it about with some judgment, the concrete can be spread evenly and with slight opportunity for the cement to get washed out from the sand and crushed rock.

In running-water, quite a different method must be followed. The cheapest is to make a more or less water-tight box, or coffer-dam, consisting of a bottom and sides. The height of the

[4] Made in the United States by the Expanded Metal Company.

sides must be slightly greater than the depth of the water. A double layer of boards having a layer of tarred roofing-paper between them makes a very suitable construction. The ground upon which the foundation is to rest is roughly leveled and the coffer-dam carefully sunk into position, by loading it with concrete. I witnessed the construction of foundations for a large stack (a very heavy spring of water having been struck in the excavation) made in this manner and, so far as known, there have been no subsequent difficulties. The construction must have been much cheaper than pumping out the water while the concrete was laid in the usual manner, and cheaper than sheet-piling.

Shrinkage.—In places where concrete is intended to finish accurately to a definite level, the shrinkage must be taken into account. The shrinkage varies with the coarseness or fineness of the ingredients, the quantity of water used in the mixing, and the rapidity of setting. Fine material used very wet and setting rapidly is the combination which gives the greatest amount of shrinkage.

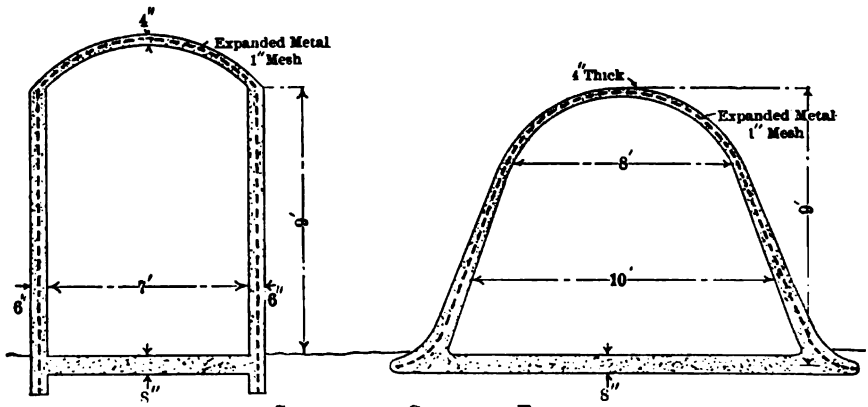
An allowance of one-eighth or three-sixteenths of an inch per foot in height for vertical shrinkage is a fair average, but great caution should be used not to exceed the desired level, as it is much easier to "shim" up the work to be placed on the concrete than it is to shave off a fraction of an inch. After the first piece of work has set, the operator has a standard of shrinkage which can be used for all of the construction, provided the materials remain unchanged, or as long as the season of the year permits an approximately similar period for the concrete to set.

Concrete Structures.

Flues.—The construction of concrete flues of the section shown in Fig. 4 gives better results than that shown in Fig. 3, being less liable to collapse. It costs somewhat more to build owing to the greater complication of the crib, which, in both cases, consists of an interior core only. For work 4 in. in thickness and under, I recommend the use of rock or slag crushed to pass through a 1.5-in. ring. Although concrete is not very refractory, it will easily withstand the heat of the gases from a set of ordinary lead- or copper-smelting blast-fur-

naces, or from a battery of calcining- or roasting-furnaces. I have never noticed that it is attacked in any way by sulphur dioxide or other furnace-gas.

Dust Chambers.—Shapes the most complicated to suit all tastes can be constructed of concrete. The least suitable design, so far as the construction itself is concerned, is a long, wide, straight walled, empty chamber, which is apt to collapse, either inwards or outwards, and, although the outward movement can be prevented by a system of light buckstays and tie-rods, the tendency to collapse inwards is not so simply controlled in the absence of transverse baffle walls. The tendency so far as the collection of mechanical flue-dust is concerned, appears to be



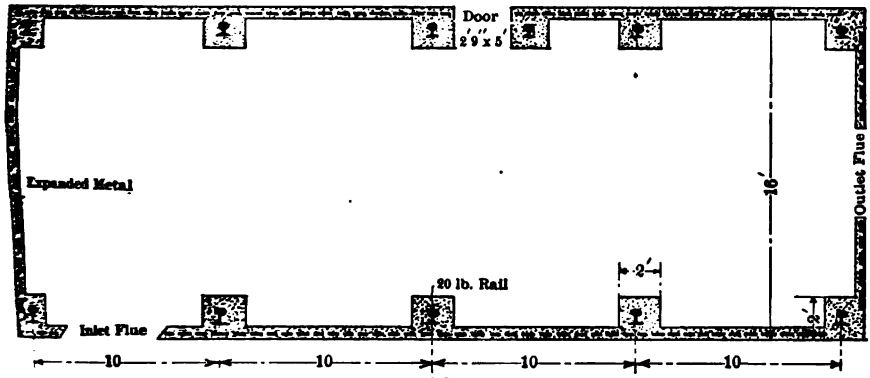
SECTIONS OF CONCRETE FLUES.

towards a large empty chamber, without baffles, etc., in which the velocity of the air currents is reduced to a minimum, and the dust allowed to settle. In the absence of transverse baffle walls to counteract the collapsing tendency, it seems best to design the chamber with a number of stout concrete-columns at suitable intervals along the side- and end-walls—the walls themselves being made only a few inches thick with woven-wire screen or "expanded metal" buried within them. The wire skeleton should also be embedded into the columns in order to prevent the separation of wall and the columns. This method of constructing is one that I have followed with very satisfactory results as far as the construction itself is concerned.

Figs. 5 and 6 show a chamber designed and erected at the Don

Guillermo Smelting Works at Palomares, Province of Murcia, Spain. Figs. 7 and 8 show a design for the smelter at Murray Mine, Sudbury, Ontario, in which the columns are hollow, thus

FIG. 5.

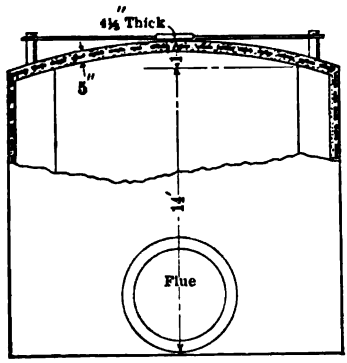


Horizontal Section

CONCRETE DUST-CHAMBER AT THE GUILLERMO SMELTING WORKS, PALOMARES, SPAIN.

economizing concrete material. For work of this kind the columns are built first and the wire netting stretched from column to column and partly buried within them. The crib is then built

FIG. 6.

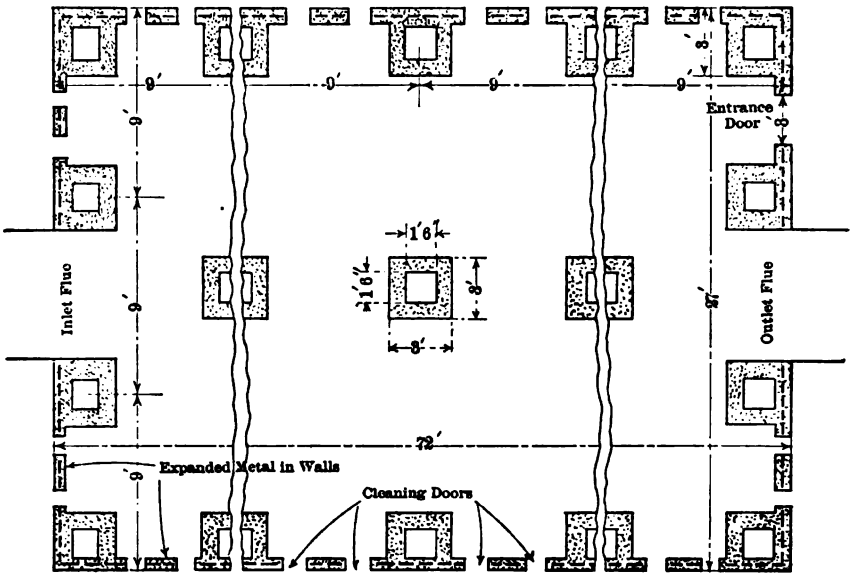


End Elevation

CONCRETE DUST-CHAMBER AT THE GUILLERMO SMELTING WORKS, PALOMARES, SPAIN.

on each side of the netting, a gang of men working from both sides, and is built up a yard or so at a time as the work progresses. Doors of good size should be provided for entrance

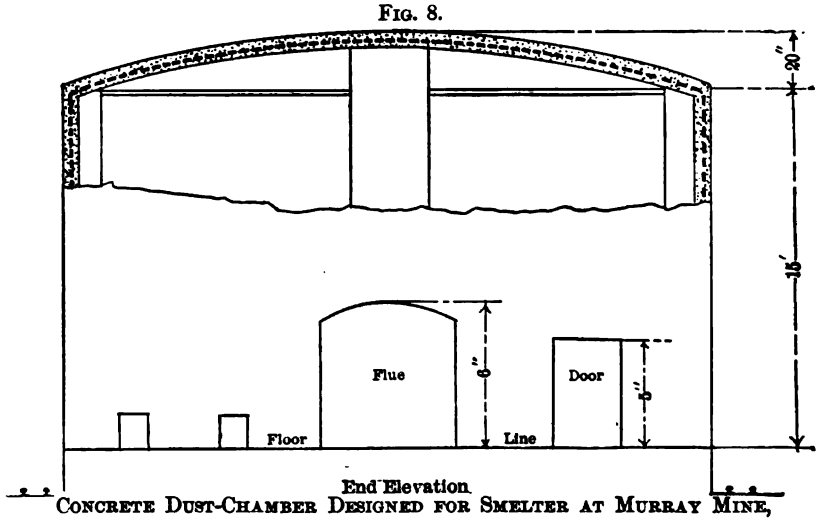
FIG. 7.



There are Eight 9-ft. Sections in the Plan

CONCRETE DUST-CHAMBER DESIGNED FOR SMELTER AT MURRAY MINE, SUDBURY, ONTARIO, CAN.

FIG. 8.



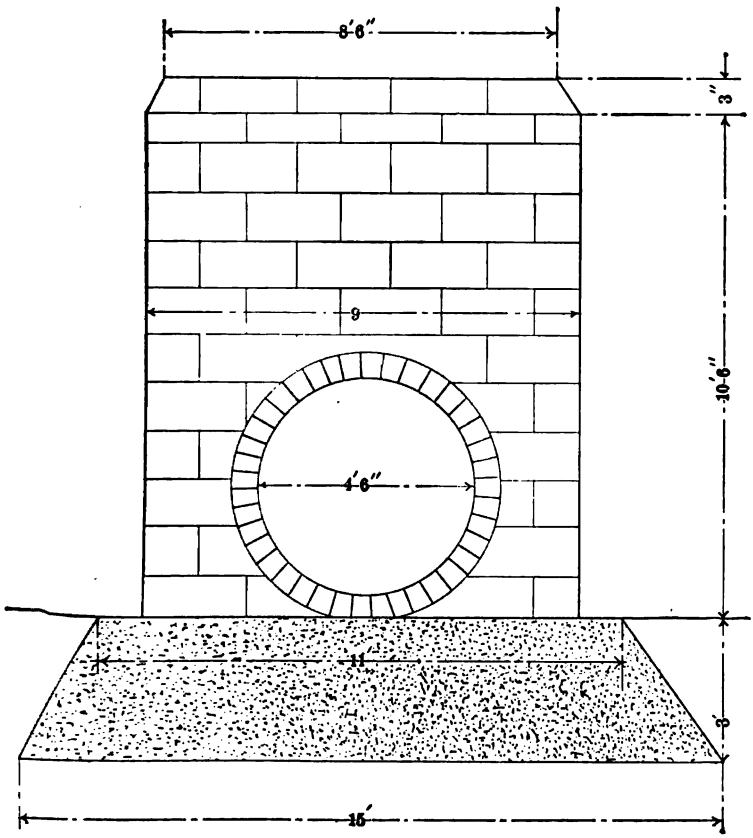
End Elevation.

CONCRETE DUST-CHAMBER DESIGNED FOR SMELTER AT MURRAY MINE, SUDBURY, ONTARIO, CAN.

into the chamber, and as they will seldom be opened there is no need for expensive fastenings or hinges.

Foundations for Dynamos and other Electrical Machinery.—Dry concrete is a poor conductor of electricity, but when wet it becomes a fairly good conductor. Therefore, if it be necessary to insulate the electrical apparatus, the concrete should be covered with a layer of asphalt.

FIG. 9.



CONCRETE BASE FOR A 90-FT. CHIMNEY AT THE GUILLERMO SMELTING WORKS, PALOMARES, SPAIN.

Chimney Bases.—Fig. 9 shows the base for the 90-ft. brick-stack at Don Guillermo. The resemblance to masonry is given by nailing strips of wood on the inside of the crib.

Retaining-Walls.—Figs. 10, 11 and 12 show three different styles of retaining-walls, according to location. These walls are shown in section only, and show the placing of the iron reinforcements. Retaining-walls are best built in panels (each

panel being a day's work), for the reason that horizontal joints in the concrete are thereby avoided. The alternate panels should be built first and the intermediate spaces filled in afterward. Should there be water behind the wall it is best to insert a few small pipes through the wall, in order to carry it off; this precaution is particularly important in places where the natural surface of the ground meets the wall, as shown in Figs. 10 and 11. If a wooden building is to be erected on the retaining-wall, it is best to bury a few 0.75-in. bolts vertically in the top of the wall by which a wooden coping may be secured

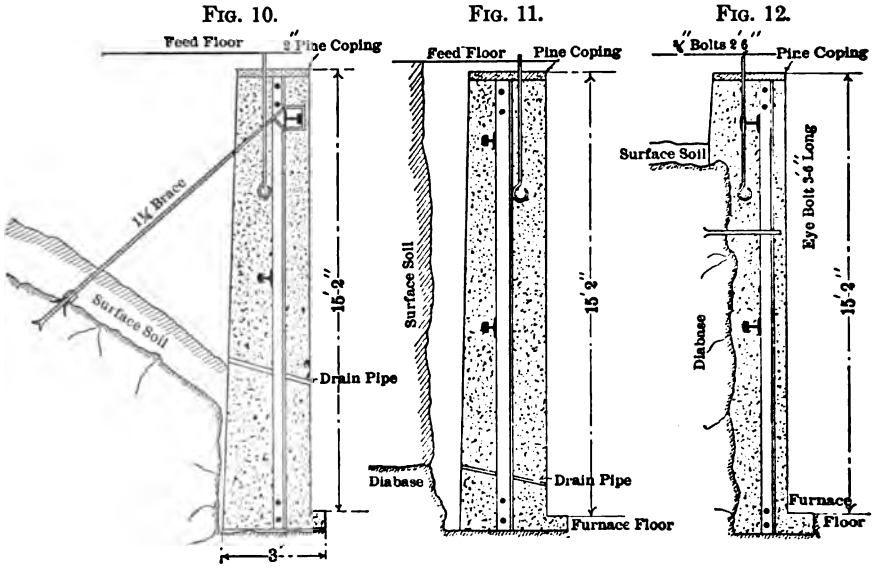


FIG. 10. FIG. 11. FIG. 12.

RETAINING-WALLS OF CONCRETE.

(see Figs. 10, 11 and 12), which forms a good commencement for the carpenter work.

The minimum thickness for a retaining-wall, having a liberal quantity of iron embedded therein, is 20 in. at the bottom and 10 in. at the top, with the taper preferably on the inner face. In the absence of interior strengthening-irons the thickness of the wall at the bottom should never be less than one-fourth the total height, and at the top, one-seventh of the height; unless very liberal iron-bracing be used, the dimensions can hardly be reduced to less than one-seventh and one-tenth respectively. Unbraced retaining-walls are more stable with the batter on the

outer face. Dry clay is the most treacherous material that can be had behind a retaining-wall, especially if it be beaten in, for the reason that it is so prone to absorb moisture and swell, causing an enormous side-thrust against the wall. When this material is to be retained it is best to build the wall superabundantly strong;—a precaution which applies even to a dry climate, because the bursting of a water-pipe may cause the damage. In order to avoid horizontal joints it is best, wherever practicable, to build the crib-work in its entirety before starting the concrete. In a retaining-wall 3 ft. thick by 16 ft. high, this is not practicable. The supporting posts and struts can, however, be completed and the boards laid in as the wall grows, in order not to interrupt the regular progress of the tamping. A good finish may be produced on the exposed face of the wall by a few strokes with the back of a shovel up and down against the crib.

In conclusion I wish to say that this paper is not written for the instruction of the civil engineer, or for those who have special experience in this line; but rather for the mining engineer or metallurgist whose training is not very deep in this direction, and who is so often thrown upon his own resources in the wilderness, and who might be glad of a few practical suggestions from one who has been in a like predicament.

Fuel and Mineral Briquetting.

BY ROBERT SCHORR, SAN FRANCISCO, CAL.

(Atlantic City Meeting, February, 1904.)

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Introduction.

Briquetting is apparently a very simple operation, yet it has offered so many mechanical and chemical difficulties that it required almost a century to place it on a commercial basis. Since 1860, however, the progress has been steady and satisfactory, and to-day, for most of the European coal companies, the briquette-plant is a very profitable investment.

Among the many applications of scientific knowledge to further economy by recovering and utilizing by-products and raw materials, there is possibly none which deserves the attention of Americans to a higher degree than briquetting. The recent miners' strike in Pennsylvania, which resulted in a scarce and irregular supply of coal at a high price, has advanced the possibilities for briquetting, and as a consequence public interest has grown rapidly.

While the manufacture of coal-briquettes is not new in the United States, adverse conditions and sometimes lack of

careful and scientific research have made the earlier efforts a failure.

The German and French literature on the subject of briquetting contains a great deal of information, but up to the present time comparatively little has been published in the English language. Our *Transactions*[1] contain some very able papers on the subject, and there are also two contributions in the publication named below.[2]

There are three guiding aims which form the basis of the Briquetting-Industry, viz. :—

1. The utilization of the practically worthless débris which is unavoidable in mining.

2. The creation of a good, hard fuel to burn without smoke and odor, by cleaning and compressing material of inferior heating-value.

3. The concentration of the greatest number of heat-units into the smallest space practicable.

These three items are of eminent importance in domestic life as well as in locomotive and marine practice. More than 100,000,000 tons of coal are consumed annually for domestic purposes alone, and possibly from 40,000,000 to 50,000,000 tons for locomotion on sea and land.

1. In the mining of coal it is unavoidable that a certain proportion should be fines (dust, slack, culm), and it is not unusual that more than one-half of the total output is brought to the surface in this form.

Up to the last 60 years nearly all of the fines were thrown upon the waste-heap,—a practice which is still followed in the United States, with the exception of the bituminous and semi-bituminous fines that are utilized to some extent in making coke. There are huge mountains of coal-slack and anthracite-culm around Eastern mines as well as vast lignite-deposits and peat-bogs in North America which have been but slightly explored as yet.

Even in countries where fuel is cheap, the waste and negligence that has been going on for a long period of years has become generally recognized.

Patents have been taken out for the manufacture of bri-

[1] *Trans.*, vi., 214; viii., 314; ii., 143; ix., 294; xx., 115.

[2] *The Mineral Industry*, vol. vi., p. 177; vol. vii., p. 191.

quettes since the time of Queen Elizabeth. Every conceivable material has been suggested as an agglomerant to bind the small particles of coal together, and many hundreds of presses have been devised to meet the various requirements. The oldest German literature on briquetting is a publication by Herr Jars (1776) which describes the making of ball-shaped briquettes near Aix la Chapelle by the incorporation by hand of coal-fragments with loam (letten, terre grasse).

The first briquetting-works using machinery were built in 1842 at Beraud in France, and by 1867 their number was increased to 31 plants, having an annual output of over 800,000 metric tons of bituminous-coal briquettes. The present production in France, exceeding 3,000,000 metric tons, is mostly purchased by the Navy, the Messageries Maritimes and various railroad companies, but more than 500,000 tons of English and Belgian briquettes is imported annually to supply the domestic demand.

Although the developments in Germany were very slow up to the eighties that country now is in the van. In 1885 its production amounted to about 140,000 tons of coal and about 1,000,000 tons of lignite (brown coal) briquettes. To-day the Westfälische Kohlen Syndicate alone is making about 2,000,000 tons, and Germany's total yearly production—exclusive of soft peat-briquettes—approximates 9,000,000 tons.

The total manufacture of briquettes in the world in 1882 was about 4,000,000 metric tons, and it is now estimated to approach 25,000,000 tons or almost 3 per cent. of the total quantity of coal and lignite mined.

The briquetting of lignite (brown coal) is done on a very large scale near Halle, Frankfurt on the Oder, and Königsberg on the Eger, in Germany; at Marseilles in France, and notably near Cologne. At Liblar, Bruehl and Horem, there are 21 plants having 182 presses, which comprise about 26 per cent. of the total number of brown coal presses used in Germany.

I visited the Cologne district in 1902 and at that time the Donatus and the Gruhl-Hütte each made from 800 to 1,000 tons per 24 hours, while the total yearly output exceeded 1,500,000 tons. Since then, it has increased to 2,500,000 tons and will probably exceed 3,000,000 tons ere long. The briquettes contain nearly

12 per cent. of moisture and their heating-value is 5,000 calories per kilogram or 9,000 B.t.u. per pound.

It is only since the seventies that brown coal is mined there systematically, and there are now approximately 3,500,000,000 metric tons in sight. The brown coal, when thoroughly dried at 100° C., contains C, 66.5; H, 5½; O plus N, 23.5; ash 5 per cent., but as it contains in its raw state at an average 60 per cent. of moisture, its heating power is but 2,500 calories per kilogram or 4,500 B.t.u. per pound; 3 kg. of fresh raw brown coal are taken to be equivalent to 1 kg. of a medium grade of coal (13,500 B.t.u.). In 1893 only 1,000,000 tons of brown coal were mined in the Rhine district, while in 1901 it increased to 6,500,000 tons, of which about 85 per cent. was used for briquetting purposes.

It takes approximately 2.25 tons of raw material to make 1 ton of the hard, brown-coal briquettes (Darrsteine) which form a large portion of the domestic fuel in German cities and sell there in bulk for $1.75 @ $2.50 per metric ton. For industrial purposes they are not as much favored as stone-coal briquettes.

England and Belgium each produce about 1,250,000 tons and both countries export heavily. Austria-Hungary, Spain, Italy, Russia, Holland and Scandinavia are of less importance. In the last decade, a few briquetting-plants have been erected in China, Formosa, Australia and India.

Very little has been done in fuel-briquetting in the United States, and the failures of earlier attempts were chiefly due to three causes—the cheapness of good mine-run coal, the high price and limited supply of a suitable binder and the discovery of oil in sections where fuel was expensive. Liquid fuel, however, will remain of mere local importance, unless the production increases at a far higher rate. The present annual oil-supply of approximately 80,000,000 bbl., if used solely as fuel, would be hardly equivalent to 25,000,000 tons of coal or about 6.5 per cent. of the total output of coal in the United States. The residues from the oil-distillation may become of great value as a binder; in addition there is a steady increase of by-product coke-oven pitch obtainable in sufficient quantity and at reasonable price, so that in spite of the cheapness of fuel the briquetting industry, once energetically taken up by Americans, may soon reach a very important position.

Consul-General F. H. Mason's report on "German Processes and Machinery for Briquette-Manufacture"[2] will be found very interesting. He refers to the cleanliness of German cities as compared with the smoky centers of industry in this country, and attributes this beneficial effect solely to the use of patent-fuel and the scientific construction and management of furnaces in general.

The first briquetting-works in the United States were erected at Port Ewen, N. Y., and Port Richmond, Pa., and were reported as able to produce annually about 150,000 tons of anthracite-briquettes containing 8 per cent. of bituminous coal, and 10 per cent. of hard-pitch as a binder.

The briquetting-works of the San Francisco & San Joaquin Coal Co., at Stockton, Cal., were destroyed by fire a few months ago, but were rebuilt at once. This plant is the largest undertaking of its kind in the United States. The presses used are of the continuous-acting type, designed by me in 1898 under suggestions of Mr. John Treadwell and Mr. Davis. The ideas involved are not entirely new. There are two presses installed, each capable of turning out from 180 to 220 tons of 8-oz. boulets (ball-shaped briquettes) per 24 hours. Soft asphaltum-pitch and evaporated oil are used as binder and lignite with a certain percentage of bituminous coal as raw material.

The company owns its own distillery, thus having a steady supply and even quality of binder. The briquettes are very well made and have a smooth and glazy surface.

The small briquetting plant, belonging to the Contra Costa Coal Co., is of an experimental nature, but many thousand tons of cylindrical coal-briquettes have been manufactured there. A 4-plunger Williams' press is used and soft-pitch or sometimes a mixture of hard-pitch and tar is used as binder.

Some years ago the Southern Pacific Railroad Co. imported a Middleton press from England and manufactured fuel-blocks in Oakland for locomotive-use; this plant, however, has been shut down for some time.

A lignite-briquetting plant was built at Rockdale, Texas; but owing to the discovery of the Beaumont oil-fields it was never started, and the machinery was offered for sale some months ago.

There is a small briquetting-plant in Chicago, belonging to

[2] *U. S. Special Consular Reports*, vol. xxvi. (1903), p. 78.

the Chicago Patent Fuel Co., which makes egg-shaped briquettes or "eggets" on a so-called Belgian press of the Loiseau design, built by the Chisholm, Boyd & White Co. of Chicago; the wear and waste is very considerable. From 9 to 10 per cent. of hard-pitch is added as a binder.

A considerable number of coal- and lignite-briquetting plants have been built in the United States, but many of them have been unsuccessful owing to the lack of study and vague experimenting on lines which were abandoned long ago.

While I am not familiar with operative plants in the United States other than those named above, I am aware that new plants are in contemplation and that orders for briquetting-machinery are said to have been recently placed in Germany.

The principles of briquetting are now thoroughly understood and a well-established and extremely profitable industry has been established in Europe. There is no need whatever to spend money in hazardous experiments, in order to ascertain the commercial value of the process.

Brown coal (lignite) and peat are almost entirely neglected in this country, although as far as known most of the deposits are of higher heating-value than those in Europe. According to recent reports in engineering journals machinery has been purchased at Magdeburg for a large lignite-briquetting plant to be erected at Bismarck, N. D.

2. Characteristics of Briquettes.

The French Navy and the Belgian State Railway issue the following specifications for the supply of briquettes:

1. The briquette must be hard, homogenous in density and size, only very slightly hygroscopic, and it should burn almost without smoke or odor.

2. The dust and breakage caused by handling and transportation should not exceed 5 per cent.

3. The specific gravity should not be less than 1.19.

4. The briquette should ignite readily, burn with a cheerful flame, and retain its shape until completely burned.

5. The ash should not exceed 9 per cent. and the evaporation-results should at least equal those of the best lump-coal, from the screenings and dust of which the briquette was made.

6. The quantity of hard-pitch to be used as a binder should

amount to 8 per cent., and the weight of a single briquette should not exceed 10 kg. (22 lb.).

Some railroad companies do not permit more than 6 per cent. of ash, and for domestic purposes a maximum weight of 1 kg. (2.2 lb.) is often specified in order to avoid breaking up before firing.

Of the many shapes used, the prismatic shape with rounded edges is mostly the most popular. Heavy blocks allow of a large output with a comparatively small investment, and they are very convenient for storage. However, they have the disadvantage of large, smooth surfaces, and unless broken up prior to being fed into a furnace they are apt to smother the fire and to choke the draft, a circumstance which is nearly always the case with a poor grade of coal or one that has been too finely ground.

To facilitate the breaking of the large blocks, channels are pressed into the bricks, or they are perforated in one operation while being formed in the press. This construction offers the advantage of a better air-circulation. The manufacture of hollow-, tube- or polygonal-shaped briquettes is very limited. The French Navy estimates 820 kg. of fuel-blocks per cubic meter of bunker capacity (more than 51 lb. per cu. ft.), that is, 10 per cent. more as compared with the storage of lump-coal. The losses in dust seldom exceed 4 per cent., while the best Welsh coal averages about 30 per cent. and in stormy weather nearly 50 per cent. dust, which reduces the stored heating-capacity very considerably. Railroad transportation, even for long distances, causes generally not more than 0.3 per cent. of dust.

Cylindrical, ball- and egg-shaped briquettes give still less dust and breakage, but they are wasteful in space. Their shape insures a good air-circulation and consequently a complete combustion.

It is not advisable to use a poor grade of fuel for such briquettes as they ignite too slowly and it is rather troublesome to break them up. They are used solely as domestic fuel and their manufacture is on the decline.

The specific gravity of briquettes varies with the material and pressure employed, and is usually as high as that of the fuel from which they have been made, *i. e.*, from 1.1 to 1.4.

In coke-making there is usually from 4 to 7 per cent. of dust,

but while the utilization of coke-briquettes in the smelting-furnace effects considerable saving in the fuel cost, their slow combustion decreases its capacity from 35 to 40 per cent. These coke-briquettes do not burn in the upper parts of the furnace and their heat effect is almost completely concentrated right at the tuyere-zone. They sustain considerable burden without getting crushed, and for this reason less blast-pressure is required. Experiments were also made using $\frac{3}{8}$ coke and $\frac{1}{8}$ coke-briquettes, but with the same result of decreasing the furnace-capacity which offsets the economy gained otherwise.

The absolute and pyrometric heating-value of a briquette is usually higher than that of the raw material. This is mostly due to a more nearly complete combustion, as even the best binder—hard-pitch—adds only from 2 to 4 per cent. to its heating-value.

Well-made briquettes do not stick together, and stoking with them is easy for the reason that no slag is formed, thus avoiding the opening of fire-doors in order to clean the grates. Concerning their smokeless and odorless combustion, full information will be found under the heading "Binders," later in this paper.

If plain grating is used the air-spaces should be from 3 to 4 mm. ($\frac{1}{8}$ to $\frac{5}{32}$ in.), and the ratio of the heating-surface to the grate-surface should be from 30 to 40 to 1. A draft of from $\frac{1}{4}$ to 1 in. water-column is sufficient. For burning brown-coal briquettes under steam-boilers, inclined grates are preferable to plain or step-grates. The inclination of the grates should be adjustable, as in the Topf-furnaces. Automatic stoking gives very good satisfaction with briquettes.

3. The Manufacture of Coal- and Coke-Briquettes.

The size and cleanness of the fuel are important items. The grains should not be larger than 0.25 in. and not less than $\frac{1}{32}$ in. in size to make a good-burning briquette. If the coal is ground too fine it will make a very handsome-looking briquette, but it will not ignite as readily and it takes a strong draft to burn it successfully. The ash-content should not exceed 6 per cent. If greater than this amount, the coal should be washed by water or treated in a pneumatic separator in order to remove the excess of ash.

If presses with "solid resistance" are used, the raw material must be of a commercial dryness, but in open-mold presses a large amount of moisture may be present. This statement applies also to the presses used in the wet-process of making peat-slabs (presstorf) and brown coal bricks (nassteine). No binder is used with the two last-named styles of presses, which, however, are of almost no importance in stone-coal briquetting.

There is no doubt that any dry-material can be bricked without the aid of a binding substance, but the high pressure and consequently the large amount of power usually necessary makes a financial success impossible.

It does not pay to briquette fat, bituminous coals which do not require a binder; although brown coal and peat briquettes can usually be formed without a binder. In the latter cases the high frictional heat-development in open-tube presses is sufficient to liquefy the bitumen of the raw-matter which acts as a binder.

Many attempts have been made to adopt a similar method for stone coals, and the earliest patent in this direction was granted to Sir Henry Bessemer in 1850. He used an open-mold press and the briquettes were very hard, but their cost was too expensive. Baroulier's patent of 1855 for heating coal in closed vessels without a binder before pressing proved also of no commercial value.

In Germany anthracite is scarce and briquettes made from it are of little importance. There is no difficulty in briquetting anthracite-culm with the addition of pitch only, but in some locations an additional mixture of a few per cent. of bituminous coal will be found advisable.

The addition of bituminous coal to the raw material imparts coking-properties which are very essential. The chemical preparation of lignites, as has often been tried, will never be profitable.

At the present time, stone coals only, but these almost always, are briquetted with a binder.

Binders.

A desirable binder should increase the burning and calorific qualities of the briquetting material, it should be free from ash and it should make a solid and waterproof briquette. Organic substances being combustible are preferable to inorganic materials which necessarily add to the ash of the fuel.

Organic Binders.—Stone-coal tar and the products of its distillation—various grades of pitch—are mostly used for a binder. In European practice hard-pitch is of foremost importance, and when of good quality it should contain from 75 to 80 per cent. of carbon and only from 0.25 to 0.5 per cent. of ash, a composition which corresponds with about 8,000 calories in heating-value. The addition of from 5 to 10 per cent. of pitch as a binder improves the heating-value of fuel from 2 to 4 per cent., depending on the number of heat-units possessed by the raw material. The utilization of tar and soft-pitch was of passing importance only, as it offers too many disadvantages. The presence of the light and the lower heavy carbo-hydrates creates smoke and smell, and the briquettes have to be kept cool, otherwise they will soften and stick and form big lumps which are troublesome to remove from the bins on land or bunkers of the steamers.

The distillation-point of soft-pitch is about 400° F., while that of hard-pitch approximates 800° F. The specific gravity of hard-pitch lies between 1.2 and 1.8; it is brittle and for use as a binder it is usually ground fine and mixed cold with the coal-slack. The mixture passes through a drier and thence to the press. In some Belgian plants hard-pitch and from 1 to 1.5 per cent. of tar are used, but briquettes of this kind burn with smoke and odor.

For pitch-making coal-tar is mostly used; the tar from brown coal, pitch and bituminous shale being of more value in the manufacture of mineral oils and paraffin-paints. The tar of wood and petroleum is of local importance only.

The coal-tar is obtained as a by-product in gas- and coke-making and to a limited extent from furnace-gases in iron works which use raw-coal as a fuel.

Tar as a by-product in coke-making was discovered by Herr Stauf at Fischbach in Germany, 1768, but by-product coke-ovens have been built on a large scale only in the last 25 years. Coal yields from 3 to 6 per cent. of tar, depending on its composition, and in distilling the tar about 66 per cent. of hard-pitch is obtained. Estimating that Europe will soon reach the 15,000,000-ton mark in the production of stone-coal briquettes, and assuming that 7 per cent. of hard-pitch will be used as binder, more than 1,000,000 tons of hard-pitch will be consumed. To obtain this quantity almost 1,600,000 tons of tar

will have to be distilled which corresponds with the coking of about 50,000,000 tons of coal.

The price of pitch being rather high (from 30 to 40 francs in France, 40 marks in Germany, from 32 to 40 shillings in England), the cost of the binder is the most serious item in the manufacture of pitch-coal briquettes, and a cheaper substitute was sought for many years. No other organic or inorganic substance, however, has yet attained more than local importance as a binder, and of those that have been used the following are the most important.

Dextrin or starch may be used. Starch or amylin heated with water becomes sticky, especially if a weak solution of sulphuric acid be added. Spoiled rye-flour, potatoes, Carragheen-moss (Cetraria Islandica), etc., contain starch and can thus be utilized. The first patent for such a process was granted to John Piddington in 1858, who used 36 lb. of flour and 8 per cent. of water for each ton of briquettes. The briquettes so made have to be heated in order to harden them. If gypsum or soft-pitch is added no heating is required.

Vegetable and animal albumin are little used. Seaweeds in a state of decomposition and cellulose make hard briquettes, but they absorb moisture too rapidly. This objectionable feature may be prevented by adding 2 per cent. of pitch or water-glass.

Sugar-molasses, as a binder, has become important since the patent was granted to Saltery in 1884. Briquettes with this binder possess nearly all of the good qualities of those bound with pitch, but they are very hygroscopic. The addition of a small percentage of linseed-oil remedies this fault. From 1 to 1.5 per cent. of molasses dissolved in hot water is sufficient for the purpose of binding, and as the molasses contains only 10 per cent. of ash it increases the ash-content of the fuel only from 0.1 to 0.15 per cent. The binding quality is due chiefly to the pectin and to a less extent to the sugar, both raw-molasses (osmirte molasses) and molasses still containing sugar (unosmirte molasses) are good materials for the purpose. The price of molasses in Germany and Austria permits a considerable saving in the expense of the binder. The process itself is very simple, but the briquettes have to be dried to become sufficiently hard to permit handling.

It is advisable, in my opinion, to use the hard-pitch if its cost is one-half that of raw molasses.

Other organic binders, of little importance, are gum Arabic, resin, glues, oils and natural asphalt.

Inorganic Binders.—The use of inorganic binders is to be avoided wherever organic binders may be had in sufficient quantity and at reasonable cost. In tropical countries only are inorganic binders preferable, because the supply of organic substances is limited and costly and its handling usually very troublesome.

Clay, alum, sal ammoniac, milk of lime, gypsum, copper sulphate, etc., are of less importance than water-glass, which was first tried in France as early as 1858. From 0.75 to 1 per cent. of water-glass added as a binder to coal-fines makes a good briquette, but it takes from 5 to 8 days of drying before they become solid. The dry water-glass contains from 60 to 66 per cent. of SiO_2 and from 34 to 40 per cent. of K_2O or Na_2O. The mixture of the sodium and the potassium water-glass, called double water-glass, may also be used.

The mixture of sodium water-glass and clay as a binder was patented in 1874 by J. Deere in England.

The most important inorganic binder is the cheap and abundant magnesia cement, which has been known since 1855 and used as a paste and binder for making artificial teeth and stone. Magnesia cement was recommended by Dr. A. Gurlt for fuel-briquetting in 1880, and the use of 5 per cent. of this material will result in a stronger briquette than that made by any other binder. One-half of the cement is hydrate water, consequently the quantity of ash added amounts to but 2.5 per cent.

The process of using magnesia cement is very cheap as no drying is required and the only fuel expended is that for power. The briquettes harden gradually at ordinary temperature and after from 6 to 10 hours are strong enough to be stored or handled. In a few days they are capable of standing a pressure of from 7,000 to 22,000 lb. per sq. in.

A good magnesia cement consists of 30 parts of 45-per-cent. magnesium chloride ($MgCl_2$), 30 parts of 93-per-cent. magnesium oxide (magnesia) and 60 parts of water. The coal should not exceed 0.25 in. in size and should be mixed cold with the dry magnesia; then the magnesium chloride should be dis-

solved in water and thoroughly incorporated. The whole mixture should be pressed as dry as possible, and a pressure of from 150 to 250 kg. per sq. cm. (from 2,000 to 3,700 lb. per sq. in.) is advisable.

Germany derives an enormous supply of magnesia cement as a by-product of the Stassfurt mines and the potassium-salt factories.

Wherever good hard-pitch briquettes are in the market, it will be difficult for a magnesia cement briquette to compete with it on account of the higher ash-content of the latter.

MIXING.—The more thorough the mixing of the binder and the coal, the less binder will be required, provided that the mechanical character of the coal remains the same. Even very high pressure exerted in the press cannot offset poor mixing. The mixing is done cold or hot, depending on the binder used. The old practice consisted in using vertical cylinders, 3 ft. in diameter and 13 ft. high, containing a revolving shaft and beaters. Horizontal mixers, varying from 12 to 65 ft. in length, are more efficient. If hot mixing is necessary, direct-fire, superheated-steam or hot-air is used. With live-fire the results are too irregular. Superheated-steam is preferred because it is the most economical and safest. The mixer-housing is jacketed and a series of small holes permits the steam to enter into the mass.

The use of superheated-steam requires about 10 lb. of fuel per ton of briquettes which, at $6 per ton of coal, is equivalent to a cost of 3 cents. The saving in binder effected by its use varies from 1 to 2 per cent., and with hard-pitch at $12 per ton, this would be from 12 to 24 cents per ton of briquettes.

The mass in the mixer should be under steam for from 7 to 10 minutes at least, and if hard-pitch is the binder, it should have a temperature at the press of from 200 to 230° F. If soft-pitch is used the temperature should not exceed 100° F.

Any dry and hard binder can be crushed and mixed with the fuel in a cold state and the mixture passes through a drier if necessary, as is the case with hard-pitch. Sometimes, however, the hard-pitch is melted separately, as is the practice with soft-pitch or asphaltum. When the pitch is melted, the coal first passes through a drier and meets the binder in the mixer, to which both have been fed in proper proportions by a measuring device.

If the coal is of a commercial dryness, it is not necessary to do more than to heat it to at least 150° F. in order to prevent the binder from becoming chilled. There are a great many designs on the market of mixers and driers, but the exigency of limited space precludes a full discussion of them in this paper.

Brown coal and peat are briquetted as a rule without a binder by two distinct processes, one the " dry " and the other the " wet " process. In the dry process the dry sorted material is cleaned, disintegrated, and passed through a drier; it then passes direct into the hopper of an open-mold press, and forms the hard briquettes called " Darrsteine." On leaving the press they are very hot and they are conveyed for a considerable distance to cool them before storing or stacking.

In the making of brown-coal and peat-briquettes by the wet process (nassteine) the mixer and the press are combined in one machine. The material is in a semi-wet state and it is mixed and forced through a mouthpiece as is described in the following section devoted to presses.

PRESSES.—Next in importance to the mixing-device is the press. To obtain a solid briquette, it should be of uniform density, and this can only be effected by using a high pressure and by keeping a proper ratio of the cross-sectional area of the briquette to its height.

If the pressing is done against a solid resistance and from one side only, a comparatively higher pressure must be exerted and even then the density in various layers will differ.

By experiment I have found that in a block of 12 in. by 6 in. in area and 5 in. high, using a pressure of 1,500 lb. per sq. in., the density at the piston side was 1.25, at the lower end was 1.15 and in the center parts was from 1 to 1.1.

The larger the briquette the higher should be the pressure per square inch. On soft coal from 80 to 100 kg. and on hard coal from 140 to 300 kg. per sq. cm. (about 1,200 to 4,500 lb. per sq. in.) will make a good briquette for bricks up to 80 sq. in. area. For small briquettes of less than 15 sq. in. area one-half of these pressures will suffice, and the cylindrical-shape will permit of a comparatively greater depth. These conditions are governed by the frictional surface of briquette, that is, the surface in touch with the mold. Even the largest and heaviest fuel-blocks should not exceed 5 in. in depth.

The power necessary per ton of briquettes made in one hour varies, depending on the pressure used and the size of briquette made.

In presses with solid resistance it takes, for small briquettes, from 1 to 2 h.p., and for large ones from 2 to 3.6 h.p. In the same style of presses, when using direct hydraulic- or steam-pressure, more than twice as much power is consumed.

In continuous-acting presses of the open-mold type from 4 to 8 h.p. have to be allowed for.

In presses with solid resistance, but with pressure-exertion on both sides of the briquette, from 15 to 30 per cent. of the power is saved. Presses should be strong and simple in construction and should work without appreciable wear or vibration. They should require little attendance in order to minimize the cost of labor. To guard against obstructions (bolts, nuts, nails, etc.) or overfeed, which might wreck the press, a safety-device should be provided.

Open-Mold Presses.—Presses of the open-mold type are of very little importance for stone coal, although they are most extensively used for brown coal and peat.

The friction of the material along the mold-wall forms so great a resistance to the passage of the piston, that the frictional losses are extremely high.

Gerondeau found that only 30 per cent. of the power employed is utilized for pressing, while the remaining 70 per cent. is lost by friction in the press, mechanisms and molds. The heat-development is so great that the bitumen of the coal becomes liquefied. The use of a binder increases the friction and usually prohibits the use of these presses.

During the time that the piston is retiring new material falls into the mold which is pressed against the briquette previously made; thus forming a continuous moving column. This class of press, while wasteful in the consumption of power, permits briquetting of partially dry raw materials (as a rule from 15 to 18 per cent. of mechanical moisture is present), while with presses having a solid resistance, a commercial dryness with not more than 7 per cent. of mechanical moisture is necessary. Considering the low value of brown coal and peat and their high water-content, the elimination of all, or nearly all, the moisture is too expensive.

As a rule brown coal and peat contain sufficient bitumen to permit the use of open-tube presses without a binder.

The first press of this class was patented by Morcan-Devinck, in 1846, and, in 1850, another design was made by Bessemer.

For briquetting stone-coal, only the Evrard and the Evrard-Bouriez presses are used, and to a limited extent. Both permit the use of a binder. The briquettes are discharged in a continuous band, but are easily separated by various automatic means. They are very hard and have a specific gravity of 1.3. Presses making from 22 to 30 strokes per minute require 8 h.p. per ton of briquettes made per hour.

At the Krupp works the first installation using a Bouriez-press and making 5 tons of briquettes per hour, cost about $11,000. The cost of the Evrard press alone is about $7,000.

For briquetting brown coal the Exeter press, used first by the inventor to briquette peat in Bavaria, is now in general favor, excepting the few works which still make wet briquettes (nassteine) using the Hertel-Schmelzer machine. The disintegrated and cleaned material is fed into a mixing conveyor and forced through a mouthpiece as an endless band, and cut up in desired lengths, as is done in the manufacture of building-bricks. In looking up patent records I have found that the first patent in the United States, No. 55,565, for an open-mold press was granted June 12, 1866, to E. Weissenborn, and on August 28, 1866, a patent, No. 57,489, was issued to L. Elsberg for a similar press.

All open-mold presses are continuous-acting and are usually mounted with the engine on a common bed-plate. The largest machine makes 90 to 95 rev. per. min. and produces from 60 to 80 tons of 1-lb. briquettes per 24 hours at an expenditure of from 50 to 60 h.p.

Solid-Resistance Presses.—Presses with solid resistance are divided into tangential and plunger-presses. The first tangential presses, sometimes called form-wheels as well (the Milch, Fischer, David and Charlotte presses, covering the period from 1846 to 1859), consisted of two vertical wheels working against each other, carrying molds, or molds and corresponding teeth, on their peripheries. The briquettes were poorly and unevenly pressed, the waste of material and the wear upon the machine was very high, from 9 to 10 per cent. of binder was required,

and the power consumed was from 4 to 6 h.p. per ton per hour. The presses of Loiseau (Port Richmond, near Philadelphia), U.S. Patents Nos. 147,665 and 167,913, 1874 and 1875,. gave much better results.

The improved Bilan press made by Schichtemann & Kraemer, Dortmund, is more satisfactory than the Loiseau. It has four beveled-off wheels that work against each other crosswise.

A few presses of the Loiseau type were manufactured by the Chisholm, Boyd & White Co., Chicago, and I am of the impression that the Chicago Patent Fuel Co. still uses one, although the expense in roller-shells is very high and the briquettes are very poorly made. This last-named company makes egg-shaped briquettes (eggets) at Chicago. Almost a dozen presses of this type have been patented in the United States, but in Europe they are of little importance. I recently designed a tangential press which will make about 2,000 small cylindrical briquettes per minute. There are a number of good features in the machine and it will be tried shortly.

While tangential presses are continuous-acting, nearly all plunger-presses are intermittent in action. The plunger-press is very simple in principle, and the more important ones are included in the following descriptions:

The Mazeline press using direct steam-pressure is still in use in France and has been recently operated at Port Ewen in the United States. The bricks formed in this type of press were 10 in. by 6 in. by 4 in. in size, and as a pressure of only 600-lb. per square inch was exerted, 10 per cent. of hard-pitch had to be used as a binder.

In spite of the use of a direct pressure some of these presses work quite economically and only about 2 h.p. is necessary per ton of briquettes per hour. The consumption of steam is about 13 kg. per ton of briquettes produced.

The press of Dupuis et Fils allows the pressure to be regulated as well as the shaping of the briquettes, the changes being effected in from 4 to 5 minutes only.

Their machinery for a plant making 5 tons of 1-kg. briquettes per hour costs about 17,400 francs exclusive of the power-installation.

Among the presses using indirect pressure are the Middleton-Detombay, the Yeadon, the Uskside Engineering Works,

the Couffinhal, the press of Dupuis et Fils, Paris, of Schichtemann and Kraemer, Dortmund (a variation of Couffinhal) and Zeitz.

The Couffinhal press, which allows the manufacture of various sizes by simply interchanging the form-table, is steadily gaining in importance. A safety-device prevents the pressure from exceeding 300 kg. per sq. cm.

A plant producing from 5 to 7.5 tons of briquettes per hour employs 5 men and 4 boys as a rule for loading or stacking. More than 200 Couffinhal presses are used in European plants, each capable of making from 120 to 360 tons of briquettes per 24 hours, depending on their size.

In 1887 the aggregate maximum capacity of about 80 plants equipped with Couffinhal presses amounted to 3,500,000 tons per year.

Presses with hydraulic pressure, as the Revollier, Rodmer and Roux-Veillon types, are comparatively little used. The Revollier press makes from 20 to 40 small briquettes per stroke, or 200 tons per 24 hours, requiring 60 h.p. to operate.

Presses with solid resistance are the most economical wherever a binder has to be employed. As a rule, the cost of fuel for steam-power is from 2 to 2.25 per cent. of the total cost. The data regarding the power necessary for pressing have been given on p. 96.

Among American presses of this design the B. C. White press, U. S. Patent No. 695,773 (March 18, 1902, patent filed June 29, 1901) and the Schorr press, U. S. Patent No. 667,254 (February 5, 1901) offer novel features. Modern briquettes seldom give more than 0.3 per cent. of dust when transported even for considerable distances.

Cost of Briquetting and Sales Prices.—In briquetting, as well as in every other industrial undertaking, the cost of the plant, land, power, labor and raw materials governs the cost of production. In briquetting, the size of briquettes, class of press and nature of binder also have a strong bearing upon the cost of the first installation. The smaller the briquette the higher will be the investment per ton of product. The whole machinery for a briquette-plant using, for instance, a Couffinhal press making 5 tons of 3-kg. briquettes per hour, with 200 kg. pressure per sq. cm., costs about $10,500 which, at 5 per cent.

interest and 10 per cent. allowance for depreciation, would equal a cost of about 4.5 cents per ton of briquettes. Taking a Couffinhal plant of a capacity of 7.5 tons of 5-kg. briquettes with a pressure of from 240 to 300 kg., the machinery costs about $14,000, which is equivalent to a cost of about 4 cents per ton, based on continuous working. The cost of erection and buildings, etc., in both cases is about the same. The power for the press and the auxiliaries per ton of briquettes made per hour varies from 4 to 8 h.p.

The cost of labor per ton of briquettes in modern plants is in France from 0.5 to 0.75 franc, in Germany 50 pfennigs, in England from 5 to 9*d.*, and in Austria 33 kreutzers, or an average of 12 cents per metric ton.

The cost of fuel, lubricants, stores and office expenses per ton of briquettes amounts in Germany to about 0.85 mark, in France to from 0.4 to 1 franc, while for interest and depreciation the cost in France is from 0.3 to 0.5 franc, and for binder from 2 to 5 francs, if hard-pitch is used.

The total cost (exclusive of binder) of briquetting 1 ton, thus varies from 1.2 to 1.7 francs in France and from 0.9 to 1 mark in Germany. The average cost in England is 1 shilling (24 cents). The cost of binder forms, as a rule, from 60 to 70 per cent. of the total manufacturing cost.

The sales-price of briquettes is entirely governed by the prices and quality of coals against which it has to compete. As a rule, it is from 66 to 80 per cent. of the price of the best lump-coal, although in some places it even exceeds a somewhat higher limit. Much depends on the absolute heating-value of the briquettes, for the reason that even the smallest consumer, after looking over his fuel-bill, will find out whether it is to his advantage to use briquettes at a certain price or not.

Coal-briquettes sell in Germany at $2 @ $3, and brown-coal briquettes at $2 @ $2.50 per metric ton (about 10 per cent. less for the short ton). In making preliminary figures it will be safe to assume 5 cents per ton for interest and depreciation.

For western American conditions the following estimate on the cost per ton of briquettes is very nearly correct.

TABLE I.—*Estimated Cost Per Ton of Briquettes in Western America.*

Labor, inclusive of stacking,	$0.16
Oil and grease,	0.006
Sundry stores,	0.01
Steam-fuel,	0.04
Depreciation,	0.05
	$0.266
8 per cent. of pitch at $12 per ton,	0.96
1,840 lb. of coal-slack at $1 per ton,	0.94
Total cost per ton of briquettes,	$2.166

In the East, coal-slack is almost worthless and the cost of briquettes will be even lower than $2.166 per ton.

4. THE MANUFACTURE OF PEAT-BRIQUETTES.

The statistics of the peat-fuel industry are not reliable, the business having been conducted to a large extent by individual enterprise, mostly in the rural districts of Germany, The Netherlands, Scandinavia, Ireland and Russia. The use of peat as fuel in Germany dates back to the earliest history of the Teutonic tribes.

The peat-bogs cover very extensive areas in the northern temperate regions of Europe and America. The German peat-area is estimated to be about 30,000 sq. km. (more than 11,000 sq. miles) and the yield per sq. km. for a depth of 5 m. (16 ft.) is approximately 800,000 tons of air-dried peat. These figures illustrate the enormous amount of stored fuel in those bogs, which, in some localities, notably in Ireland, reach a depth of 15 m. (48 ft.).

Peat is utilized for hygienic purposes, for manufacture into paper-stock, cardboard, felt, alcohol, etc., for burning in gas-generator furnaces and for manufacture into peat-coke, peat-slabs and carbonized-peat briquettes. The application of pressing or briquetting pertains solely to the slabs and briquettes and for this reason their discussion only is taken up in this paper.

Solid Peat-Slabs (Machine-Peat).—The manufacture of peat-slabs has been practiced in a crude way by peasants, in the north of Germany and in Holland, for more than a century, for the purpose of obtaining a cheaper and more efficient fuel than wood or cut peat (stichtorf). During the past 50 years this

industry has been placed on a more intelligent basis due chiefly to the efforts of C. Schlickeysen, A. Vogel, Exeter and Hebert. The problem of a cheap production on a large scale has been solved, and to-day machine-peat is made which stands transporting and the influences of weather, and, in many localities, even competes with coal.

The machines of Schlickeysen, Mette-Sander and Andry are mostly used, each having been designed to fit an individual case for the reason that the physical properties of peat, and its occurrence as peat-bog, are quite different. In heating-value, 100 kg. of medium-grade coal are equal to about 160 kg. of machine-peat or to 222 kg. of raw-peat, but the great drawback to raw-peat is that it requires between three and four times as much storage space as coal requires.

The method of making machine-peat is entirely automatic, the machinery for cutting the peat, elevating it to the press and conveying the slabs to the drying-grounds being mounted on a truck which travels into the bog sometimes under its own steam. This apparatus is made for a capacity of from 50 to 80 tons in 24 hours and costs from $4,000 to $6,000 at factory. The truck travels on rails and the bog is gradually exhausted by cutting each new trench next to the one just completed.

An excavating elevator drops the raw peat into the machine where it is disintegrated, kneaded and forced through a mouthpiece, in the form of an endless, plastic band, on to a conveyor upon which it is cut into any desired length by a series of adjustable knives. The pressure required is very slight and, as no water escapes, the chemical composition of the raw material is unchanged. The volume of the peat is reduced about one-half, and the slabs when thoroughly air-dried weigh from 40 to 60 lb. per cu. ft. One man is employed for from 2 to 2.5 tons of peat-briquettes produced.

While the raw peat contains as a rule between 80 and 90 per cent. of moisture, the air-dried slabs have seldom more than from 15 to 25 per cent. To effect a more thorough drying, large hot-air chambers are sometimes used.

The first machine for this purpose was designed in 1861 by C. Schlickeysen of Berlin.

The cost of making machine-peat is from 3 to 4 marks (from

$0.69 to $0.92) per metric ton at the outset, which allows a considerable depreciation for the machinery. This figure is taken from the Schilt works near Oldenburg, which has a capacity of 75 tons per 24 hours, and from the Ranbow works near Langen on the Elbe, which makes 50 tons per 24 hours.

Well-drained peat-bogs in the north of Germany pay at a handsome rate wherever there is a market in the near vicinity.

The peat-bog at Magdeburg yields annually about 10,800 marks' worth of machine-peat per acre, while the cost of manufacture is but 3,600 marks, thus leaving a profit of 7,200 marks per acre. The average depth of this bog is 40 ft.

The experience gained with the use of press-peat as locomotive-fuel in Bavaria, Austria, Sweden, Ireland and Russia is very satisfactory. The cleanliness of the combustible more than trebles the life of the fire-box and boiler-tubes. The use of drying-chambers reduces the moisture in the peat-slabs to from 5 to 6 per cent., which is of great advantage wherever peat has to be transported any appreciable distance; furthermore, it renders the manufacture of this fuel entirely independent of climatic influences and the plant can be kept going throughout the year. The utilization of dried press-peat for gas-making and as substitute for stone coal and charcoal is very satisfactory.

Solid, Carbonized Peat-Briquettes.—The problem to produce from a poor grade of fuel, containing from 70 to 90 per cent. of moisture, a briquette which can compete with stone coal, or can make up deficiencies in the fuel-supply, is a very serious one. Huge masses of raw material have to be handled and cleansed from foreign matter, and tons of water have to be expelled in order to obtain a limited quantity of valuable fuel. Many processes have been tried and abandoned, as they proved to be too expensive. A few plants in Germany and Holland are working on similar lines with brown coal, but a large portion of the water is expelled mechanically before drying by heat. The usually fibrous nature of the peat makes the use of centrifugals and filter-presses rather inefficient, and its low conductivity and charring-point renders the drying very slow and expensive, for the reason that the temperature used should not exceed 250° F., otherwise the composition of the peat would be affected. The operation of drying is less serious if the peat-

bog permits of a proper drainage and if the peat is brought in an air-dried condition to the drier, preferably of the Schultze or the Schorr type (U. S. Patent No. 655,418, Aug., 1900).

In recent years the experimental plants at Trebbin and Ostrach, utilizing the Stauber patents, are attracting great attention, and, according to recent publications, the experience is so satisfactory that a large plant at Stettin and another at Kõnigsberg are under construction. The calorific value of a German peat-briquette approximates 4,000 calories per kg. (equivalent to 7,200 B.t.u. per lb.), and the product is sold at $1.75 @ $2.50 per metric ton.

The Schuelke and the Schoenning-Fritz processes also are very promising, and a number of works are being constructed which will use them.

The old peat-industry was worked solely to supply the domestic demand and its existence was a "life from hand to mouth." The machinery and methods then used permitted small output and poor quality. However, slow and poor work is now being replaced by quick-working machinery.

Much labor and money have been spent in Germany on the development of the peat-industry and nearly all modern methods have originated in that country. However, in my opinion, the problem has not yet been fully solved. Great efforts are being made to establish especially the manufacture of solid peat-briquettes as a permanent commercial industry. I trust that this will be accomplished and that the treasure with which Mother Nature has endowed so many countries will be used to the benefit of mankind.

A perfect peat-fuel would soon find most extensive use. The peat-bogs near large settlements would gradually disappear and thus the sanitary conditions of the locality would be improved.

In some places the bottom of the peat-bog exposes fertile ground which could be used for agricultural pursuits if the bog were removed.

There are many acres of peat-bog excavations in Holland under cultivation and supporting from 300 to 350 people per square mile. In some water-filled bog-trenches fisheries are established on a large scale.

In conclusion, I append a few financial calculations referring

to the manufacture of carbonized peat-briquettes, for the reason that this product forms the most modern branch of peat-utilization.

Estimates of Cost of Operating Peat-Briquetting Plants.—A one-press plant. (Approximate capacity, 60 tons per 24 hours.)

The average amount of moisture in the raw material is 85 per cent. The peat may be of a fat, earthy or fibrous nature. The first question is, "How much raw material has to be raised to obtain a certain weight of briquettes?" With one press of the heaviest pattern working 24 hours per day, from 48 to 60 metric tons of briquettes are made, depending on the specific gravity of the peat. A good quality of raw peat with 85 per cent. of moisture weighs from 63 to 66 lb. per cu. ft. Assuming an average output of 55 metric tons (60 short tons) per 24 hours, to manufacture 60 short tons of briquettes, containing 15 per cent. moisture, in 24 hours, 10,800 cu. ft., or 680,000 lb. of raw peat, containing 85 per cent. moisture, must be cut at the bog, as calculated below. Deducting the 15 per cent. of moisture contained in the 120,000 lb. of briquettes, leaves 102,000 lb. of solid inflammable material which is exactly 15 per cent. of 680,000 lb.; and as 1 cu. ft. of raw peat weighs about 63 lb., then 680,000 lb. are equivalent to a volume of 10,800 cu. ft.

The next question is, "How much raw material is needed for the steam-raising purposes of the plant?"

Allowing for electric-light installation and mechanical chain-haulage from the bog, a plant equipped with one press will require a boiler-capacity of about 2,000 sq. ft. of heating-surface. Assuming the use of small, and especially non-condensing engines, we may estimate that about 8,800 lb. of water per hour would have to be evaporated. Air-dried peat containing 25 per cent. of moisture is used for fuel. If the air-dried peat is of good quality the heat from 1 lb. will evaporate 3 lb. of water per hour, and as the total quantity of water to be evaporated is 211,200 lb. (8,800 x 24), there must be burned 70,400 lb. of air-dried peat having 25 per cent. moisture, which is equivalent to 352,000 lb. of raw peat containing 85 per cent. of moisture. Hence, the total quantity of raw material needed in 24 hours is:—For briquettes, 10,800 cu. ft. (340 tons), and for steam, 5,587 cu. ft. (176 tons), making a total of 16,387 cu. ft. (516 tons), which will amount to 154,800 tons of raw peat per year of 300 days needed to make 18,000 tons of peat-briquettes.

The bog is thoroughly drained, if possible, and the cut peat-slabs are stored in heaps which are shoveled over and over again until the moisture does not exceed 25 or 30 per cent. Then large new heaps are made and from them the material is forwarded to the briquetting-plant.

At the outset the cost of obtaining one ton of air-dried peat is about 3s. (70 cents), which can be reduced to an average of not more than 2s. 4d (54 cents).

Considering the waste, it will be safer to assume 525 tons of raw peat per 24 hours necessary for the manufacture of 60 tons of briquettes. The 525 tons of raw peat containing 85 per cent. of moisture are equivalent to 105 tons of air-dried peat containing 25 per cent. of moisture, or 300 x 105 = 31,500 tons per year.

TABLE I.—*Estimate of Cost of Installing and Operating a One-Press Plant.*

	Marks.	Dollars.
Total cost of machinery for 60-ton plant,	165,000	
Cost of erecting machinery alone,	12,000	
Machinery at bog, tracks, cableways, cars, mechanical haulage, etc.,	45,000	
Foundations, buildings, sheds, boiler-settings, stacks, etc.,	35,000	
Total investment,	257,000	59,110
Yearly Operating Expenses:		
(a) Cost of 31,500 tons of air-dried peat at 3 marks per ton,	94,500	21,740
(b) Depreciation:		
(1) 7.5 per cent. on briquetting-machinery,	13,275	
(2) 15.5 per cent. on bog-equipment,	6,975	
(3) 3 per cent. on buildings, etc.,	1,050	
	21,300	4,900
(c) Salaries for superintendent and clerk,	11,000	2,530
(d) Repair parts for press, belts, stores, oil, carbons, incandescent-lamps, etc. (25 marks per day),	7,500	1,725
(e) Wages (65 marks per day),	19,500	4,485
Grand total of yearly expenses,	153,800	35,380
Daily expenses, $\frac{1}{300}$, making 60 short tons,	512.66	117.90
Actual cost of making 1 short ton,	8.54	1.97
The selling-price per ton at the works is 10 marks ($2.30).		
Daily receipts 60 x 10 =	600	138.00
Yearly receipts,	180,000	41,400
Yearly expenses,	153,800	35,380
Yearly profits,	26,200	6,020

Return on the investment of 257,000 marks, or $59,110:
26,200 × 100 ÷ 257,000 = 10.19 per cent.

This result looks rather inviting, but wherever peat-briquettes have to compete with brown-coal briquettes, the price of 10 marks per short ton is rarely possible. The brown-coal briquettes, being about 25 per cent. higher in heating-value, sell on car at factory for from 10 to 12 marks per metric ton. Taking the selling-price at 9.50 marks ($2.19), the interest on the investment would amount to 6.65 per cent., which, for European conditions, is still appreciable.

It should be noticed that the slightest increase in any item shows its enormous influence in the calculation and upon the final result, as very considerable quantities of a low-grade material have to be raised which, improved and compressed to briquettes, possess, in some locations, up to 60 per cent. of the calorific value of good bituminous coal.

TABLE II.—*Estimate of Cost of Installing and Operating a Two-Press Plant.*

	Marks.	Dollars.
Total machinery in briquetting-plant of 120-ton capacity in 24 hours,	260,000	59,800
Erection of machinery,	20,000	4,600
	280,000	64,400
Equipment of bog as before,	65,000	14,950
Buildings, stack, boiler-settings, etc.,	45,000	10,350
Total investment,	390,000	89,700
Yearly Operating Expenses:		
(a) Cost of 66,000 tons of air-dried peat containing 30 per cent. of moisture,	198,000	45,540
(b) Depreciation:		
(1) 7.5 per cent. on briquetting-machinery, *i.e.*, 7.5 of 280,000 marks,	21,000	
(2) 15.5 per cent. on bog-equipment, *i.e.*, 15.5 per cent. of 65,000 marks,	10,075	
(3) 3 per cent. on buildings, etc., *i.e.*, 3 per cent. of 45,000 marks,	1,350	
	32,425	7,456
(c) Salaries,	11,000	2,530
(d) Repair parts for press, stores, oil, lamps, carbon, etc. (50 marks per 24 hours),	15,000	3,456
(e) Wages (95 marks per 24 hours),	28,500	6,555
Grand total,	284,925	65,537

	Marks.	Dollars.
Daily expenses, $\frac{1}{300}$, making 120 tons of briquettes, . .	949.75	218.44
Actual cost per ton, ...	7.915	1.82
Taking the selling-price per ton at factory 10 marks, or $2.30, the yearly receipts for (120 x 300) 36,000 tons will be, ...	360,000	82,800
Yearly expenses, ...	284,925	65,531
Yearly profits, ...	75,075	17,269

Return on the investment: $75,075 \times 100 \div 390,000 = 19.25$ per cent.

Had air-dried peat with 25 per cent. of moisture been taken (as in the one-press plant), the investment would be still more favorable.

Assuming a selling-price of only 9.5 marks ($2.19) per short ton, the return would be as follows:

Yearly receipts, 36,000 × 9.50 =	342,000 marks or $78,660
Yearly expenditure, . . .	284,925 marks or $65,531
Yearly profits, . . .	57,075 marks or $13,129

Return on the investment, 14.63 per cent.

These calculations are based upon the manufacture of very hard peat-briquettes, using the most modern machinery, installed permanently in brick buildings.

5. The Manufacture of Mineral Briquettes for Metallurgical Purposes.

The value of briquetting mineral substances in connection with metallurgical processes and the manufacture of artificial stone masses is well understood and appreciated.

Mineral-Briquetting for Metallurgical Processes.

The mechanical nature of the material to be treated in blast-furnaces is of almost as great importance as its chemical composition. In smelting-plants there is always more or less flue-dust and raw and calcined fine ores, and sometimes fine concentrates to be treated, but the charging of such fine material into a furnace would cause trouble and irregular working and it would lessen the furnace-capacity also. In addition, a very considerable amount of flue-dust is collected at smelting-plants which has to be reworked at an increased cost. The briquetting of mineral substances causes considerable wear upon the ma-

chinery and is appreciably expensive in the costs of binder, labor and handling; therefore, many smeltermen try to avoid it.

The financial question, however, is not as serious as it may at first appear without a thorough investigation, and taking the large output of a modern briquetting-machine into consideration, the cost for repairs hardly amounts to a few cents per ton of briquetted material. The total cost depends in the first place on the cost of labor, power and binder, and in most of American smelters it varies between $0.65 and $1.25 per ton of briquettes produced.

Ordinary brick presses have been occasionally used in Europe and in the United States, to briquette the fine material, with clay or dilute sulphuric acid as a binder, but the presses were too slow and expensive for work on a large scale, and the presence of clay as a binder was usually undesirable.

The English Yeadon (fuel) press also has been used for a few decades at the works of the Carlton Iron Co. at Ferryhill, and of the Ore & Fuel Co., Coatbridge, England, and also by a few concerns on the Continent. Dupuis et Fils, Paris, furnished a few presses which are used chiefly for manganese-ores, iron-ores and pyrite. In some localities coke-dust also is added.

The making of mud-cakes is the crudest form of briquetting, but while heat has to be expended to evaporate the contained moisture, amounting to from 40 to 50 per cent., and while considerable flue-dust is made, even this crude operation is better than feeding the fine-ore or flue-dust directly into the furnace.

The only other method to avoid briquetting is to fuse the ore-fines in reverberatory-furnaces, and to add the flue-dust to the material in the fusion box, thus incorporating it with the slagged ore. This practice is used to some extent in silver-lead smelters, notably with lead silicate which is easily fused.

In connection with copper- or iron-smelters the slagging process is impracticable, for the reason that the losses are too severe and the expense of fuel is very considerable.

The patent of Saltery, granted in 1884, covering the use of sugar-molasses as a binder, gave a strong impetus to mineral-briquetting.

In mineral-briquetting a thorough mixing and kneading of the material is of the foremost importance; and if this be done properly, a comparatively low pressure will suffice to make a

good, solid briquette which, after from 6 to 8 hours of air-drying or after a speedier elimination of the surplus moisture in hot-air chambers, will be ready for the furnace-charge. A good briquette should allow transportation without excessive breakage or dust, a few hours after it has been made, and it should retain its shape in the furnace until complete fusion, so as to create as little flue-dust as possible. The briquette should be dense, otherwise it will crumble under the influence of bad weather.

In modern mineral-briquetting all these conditions are fulfilled to a limited extent only, and a great deal of experimenting is being done, notably in Belgium and Germany, to improve the process.

The two presses now on the American machinery market, the "White Mineral Press," built by the Chisholm, Boyd & White Co., of Chicago, and the "White Briquetting Machine" manufactured by the H. S. Mould Co., of Pittsburg, are extensively used. Both machines have weak points, but in most metallurgical plants it will pay well to adopt them.

Generally from 5 to 10 per cent. of milk of lime is used as a binder, which has also a beneficial fluxing-influence. A complete outfit comprises, besides the press, a mixer for slacking the lime, a feed-pump which discharges the liquid in proper proportion into the main-mixer wherein the ore-fines, flue-dust or concentrates are shoveled. The cohesive properties of slimes from the concentrators is sometimes utilized to bind the ore-fines. Mineral-briquettes are very hygroscopic, and climatic conditions are of great influence upon their hardness.

The press of the Chisholm, Boyd & White Co. makes 80 briquettes per minute, which with a new disk are 4 in. in diameter and 2.5 in. in height, thus giving about 872 cu. ft. of briquette-volume per 10 hours, equivalent to a weight of from 50 to 80 tons, depending on the material. With the wear of the disk the height of the briquettes is reduced and consequently the capacity of the machine. The disk weighs about 1,600 lb., and as most of larger smelters have their own foundry, it can be replaced with little expense. There are more than 50 of these machines in use in the United States. This press offers the advantage of bricking moist material, but it is necessary to dry the briquettes afterwards, and even then they have to be

carefully handled in order to avoid breakage and excessive dust.

To drive the apparatus 30 h.p. is usually provided. The White mineral press is too well known to metallurgists and engineers to require further comment or description in this paper.

The H. S. Mould Co., after several years of experimenting with the White briquetting-machine, succeeded in making it a success for pressing dry-material.

The White briquetting-press is of the plunger-type and is built in three sizes. The largest uses 6 plungers and at 25 revolutions makes 150 briquettes 3 in. in diameter and 3 in. in height, which corresponds to more than 1,080 cu. ft. in 10 hours.

For a plunger-type press the material must be of a commercial dryness and should not contain more than 7 per cent. of mechanical moisture. If wet concentrates are to be briquetted it is necessary to add dry ore-fines or flue-dust to them in order to obtain a proper consistency. The briquettes from this press are very solid and require air-drying only for a few hours.

The cylindrical shape of briquettes is very good for the reason that it insures a proper air-circulation and consequently a rapid oxidation and fusion.

The wear of the White briquetting-machine is mostly confined to the chilled-iron bushings and to the pistons. The auxiliary machinery consists of a slacker, a feeder and a main-mixer. The press is of a very substantial design.

Briquetting, being a crude operation, involves considerable wear and tear, but to treat flue-dust, ore-fines and fine concentrates successfully, it is usually necessary to resort to it.

Edison used a number of small intermittent-acting presses at his magnetic-iron separation works in New Jersey, but I think the plant has been shut down for some time.

In recent years a great many briquetting-plants have been added to American and European metallurgical establishments.

Ores containing a sufficient quantity of alumina can be briquetted with a so-called "stiff-mud" machine. This style of press is used very extensively for the manufacture of bricks and tiles by the wet-process. It is a rather inexpensive device and it will answer also where slimes are available to be added to ore. The endless, semi-plastic band discharging from the

machine is cut by wires and the size of slabs can be varied at will. The breakage and dust caused by transportation is but little more than that involved in handling material from ordinary presses.

Where labor is cheap, bricking by hand will be found most economical. A man can make about 1,000 bricks in ten hours, each weighing from 8 to 14 lb. There is, however, no objection to making bricks for furnace-use considerably heavier, especially if coke- or coal-fines are mixed with the ore.

6. The Manufacture of Artificial Stone-Masses.

Mineral-briquetting is also of great importance and value for the manufacture of artificial building-stones, crucibles, grindstones, art works and emery- and polishing-wheels.

The artificial masses must be solid and capable of withstanding wear, rain and frost. Many objects, which in former years were cut from stone by toilsome handwork, are obtained to-day by a simple process of casting or briquetting. By the combination of such masses with iron-constructions, buildings, bridges and viaducts of great strength and beauty are erected. Most beautiful objects of art are made in the same way, and are, in appearance, properties and color, usually equal to similar articles manufactured at high cost from natural stone.

The crudest form of artificial stone dates back to the oldest history of Egypt, when mud and clay slabs were made for building-purposes, and at the present time this process is still carried on in Mexico, South America and other tropical countries, in the manufacture of adobe.

Even the burning of clay-masses, *i.e.*, the making of common red bricks, is very old.

In countries, however, where clay or natural stone is scarce, other materials have been used. In such localities sand is usually abundant, and by the use of milk of lime and adding peat, earth, coal-ash, slag or similar material, fairly good bricks are obtained. Of greatest importance in this direction are the natural and artificial cements, which possess the property of absorbing a large quantity of sand, broken-stone, or other materials, without suffering in strength, and when mixed with water they form a stiff paste which hardens quickly and gains steadily in strength.

Gypsum also is used very extensively, but it first has to be prepared in order to prevent its disintegration under atmospheric influences.

The most important binding-substances for the manufacture of artificial stones are,—water-glass, magnesia cement, zinc chloride, gum Arabic, glue, caseine and glycerine.

The use of the most of these substances has been described earlier in this paper.

For art-goods and other industrial purposes, saw-dust, cellulose, chalk, magnesia, etc., are formed into solid masses and finished with tools similar to those used in working wood. In this manner artificial meerschaum, Roman mosaic (terrazzo), and imitations of porphyry, serpentine and marble, that sometimes deceive even the eye of an expert, are made.

Cajalith, which is used for making vases and figures, is composed of magnesia and silicic acid. It yields a clean, white product and can be colored most artistically by an admixture of dyes in small quantities.

Attractive building-stones are made by adding ocher or ultramarine in proper proportion to ash, sand and slag.

In the manufacture of slag-bricks, which forms a branch of the slag-cement industry, milk of lime or slag-cement is used as a binder, although in some cases the material can be formed into shapes without any binder whatever. Usually the slag as it flows from the furnace is granulated by a water-jet, which at the same time carries the granules forward to a settling-tank.

The casting of slag-bricks for building and metallurgical purposes is an old practice, but the pressing of the material into bricks is of a more recent date.

Considering that for every ton of pig-iron made about a ton of slag is formed, the utilization of this waste-product is of great importance. Furthermore, there are a great many lead- and copper-smelters where the disposition of slag in course of years has become extremely difficult and expensive. As a rule the laws prohibit the dumping of slag into rivers, as it would poison the fish and create obstructions in the waterway. The making of slag-bricks and slag-cement, which can be utilized to great advantage at the smelter and in surrounding cities, may solve the problem in many locations. Slag-bricks were made first at Osnabrueck in 1859, using milk of lime as a binder. At the

present time, slag-cement is largely used, and many hundred millions of slag-bricks are manufactured annually. The slag-bricks stand great pressure and heat and consequently are extremely valuable.

A mixture of broken slag (from 1- to 2-in. cube in size), slag-sand and Portland cement is called "metallik," which forms an excellent material for plastering streets, sidewalks, floors of factories, smelters and railway stations,—in fact, any place where the traffic is heavy. Steps in stairways also are made in this manner, but a mixture of cement and slag-sand only should be used for the first inch of depth in order to obtain a smooth surface.

Another class of artificial stone, made from bituminous shale and asphaltum, is extremely valuable for the construction of wet foundations, piers and docks. The asphaltum is boiled with tar; sand and sharp broken stone are added; and the mass is thoroughly mixed together. It is then brought into forms and subjected to a high pressure. The mixture consists generally of 95 parts asphaltum, 5 parts tar, 150 parts stone and 5 parts fine sand.

The mechanical operations in the manufacture of artificial stone offer no difficulties. The disintegration of the raw materials is done by jaw-crushers, rolls, Chilian- or ball-mills, centrifugal grinders of various designs and stone-mills. For the grinding of very hard stone, as feldspar or corundum, it is advisable to chill these materials first. This is done by heating the mass and dumping it while hot into cold water. The sudden change in temperature causes the stones to crack and to become more brittle, which facilitates their subsequent disintegration. There are two methods of manufacturing stone, similar to those used in fuel-briquetting, and may be distinguished as the "wet" and the "dry" processes. The making of blocks and more complicated forms in wooden or iron molds offers no special interest or difficulties.

The materials, after a thorough mixing or kneading, are ready for pressing.

In the wet process machines are used similar to those described earlier in this paper in discussing the making of brown-coal briquettes. A machine for making stone blocks, built by Hertel & Co., using about 10 h.p., has a capacity of 17 blocks per

minute, or over 10,000 in a 10-hour shift. The only labor required is feeding the machine and removing the ready finished stones. Plunger-type presses, sometimes termed Canadian brick-presses, are not so extensively used for the purpose as they require more labor. Most of the artificial stone-masses require drying or calcination before use.

Ornamental bricks from shale are also made quite extensively. In Germany large side-walk plates and roofing-slabs are made from sand and cement on a very large scale, and the manufacture of lime-sand bricks by a new method is also very successful. The sand is mixed with from 4 to 8 per cent. of lime and bricked. The bricks are placed under steam-pressure of from 105 to 120 lb. (160 to 180°C.) for from 8 to 12 hours, and thus a calcium hydrosilicate is formed, making the bricks hard and waterproof. Although this industry is hardly four years old, there are already about 100 factories in Germany using this process and more than 400,000,000 sand-bricks (kalk-sandstein) were manufactured in 1902. A factory for the manufacture of such bricks was built this year at Bakersfield, in California, and there are a number of others in the course of erection. Other artificial stone masses, however, have been made in the United States for a great many years.

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Postscript.

H. O. Hofman, Boston, Mass. (communication to the Secretary):—I should like to add to the list of books and papers consulted by Mr. Schorr, references which have been of use to me in studying the subject of briquetting.

E. T. Dumble, *Brown Coal and Lignite*, Austin, Texas, Geological Survey of Texas (1892). This book is, to my mind, the most valuable of the English publications on briquetting.

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The Equipment of a Laboratory for Metallurgical Chemistry in a Technical School.

BY CHARLES H. WHITE, HARVARD UNIVERSITY, CAMBRIDGE, MASS.

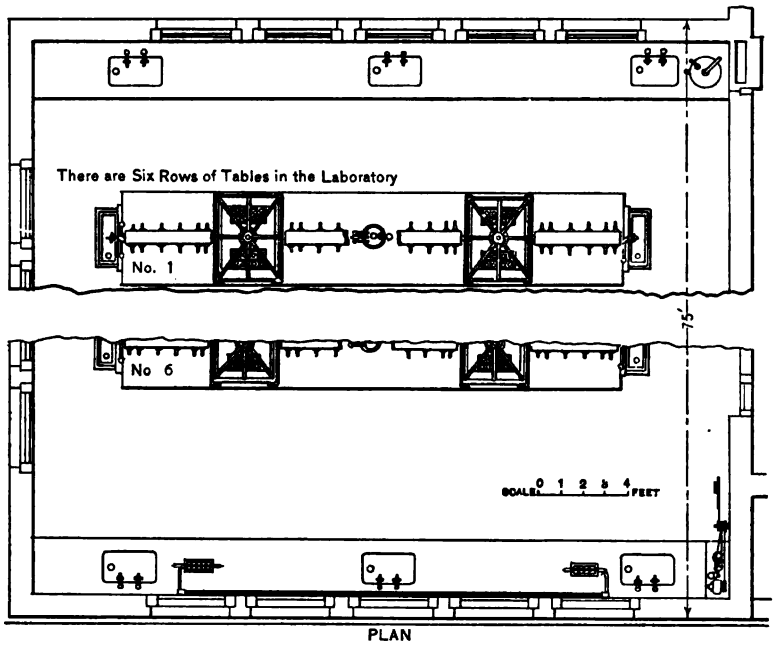
(Atlantic City Meeting, February, 1904.)

THE equipment of a laboratory in which students are to be trained for practical work in metallurgical chemistry presents many difficulties not encountered in the fitting up of a commercial or works laboratory. At an industrial plant the chemical work required usually demands the frequent repetition of a comparatively limited number of methods. In a steel-works laboratory, for instance, by far the larger part of the work consists in making determinations of the usual impurities found in iron and steel; at a smelter the great bulk of the work is confined to the analysis of ores, slags, mattes, and fuels, which involve the frequent application of perhaps not more than a dozen different methods. In a laboratory of this character, therefore, those operations most frequently applied may each have a certain section set apart and fitted permanently with such apparatus as best facilitates the carrying out of this one operation.

The work of a commercial or custom laboratory is perhaps more varied than that of a works laboratory, but the conditions are similar in these respects, that there is usually no great need for economy of space, and that a few skilled workmen are employed to carry the determinations through from beginning to end without interruption. On the other hand, the laboratory of a technical school should be planned on such a scale that a comparatively large number of students may work at the same time, and each desk should be equipped for general metallurgical analysis. In most institutions for technical education the student's program is so arranged that his day is divided among several subjects in the course of study; only a few consecutive hours being devoted to one subject, or spent in one laboratory. Therefore, those appliances should be provided that render possible the safe continuation of such processes as

precipitation and evaporation during the student's absence; in other words, full provision should be made for maintaining constant temperatures for long periods. The fixed apparatus should be so assembled that the student may carry on at the same time as many operations (evaporation, filtration, titration, etc.), as possible at his own desk, and thus minimize the passing to and fro in the laboratory. The equipment of the school laboratory, however, should conform to that of the works laboratory in this particular, that all apparatus should be selected or

FIG. 1.



PLAN

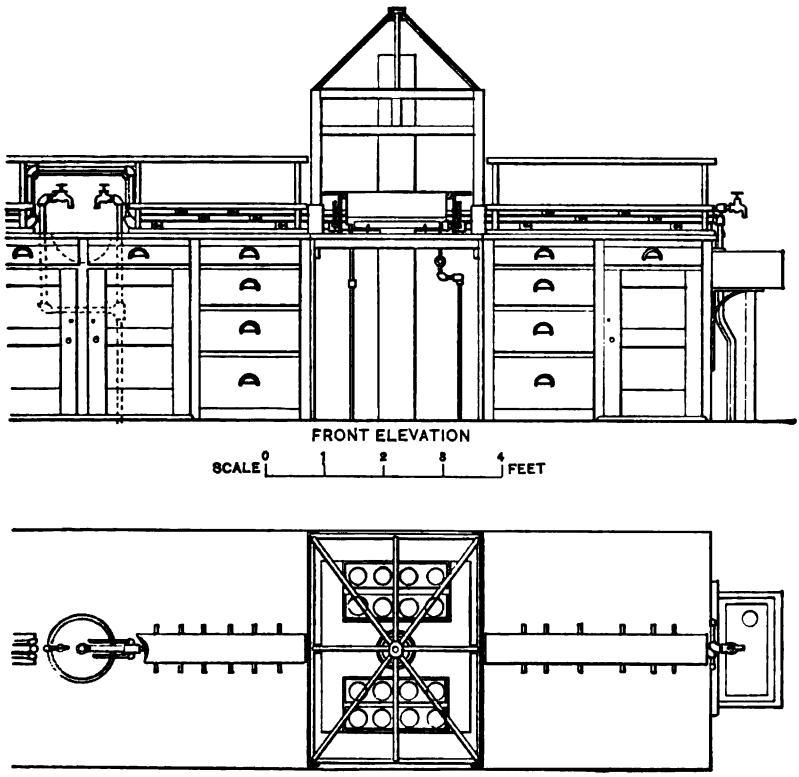
ARRANGEMENT OF WORKING-TABLES IN THE HARVARD LABORATORY FOR METALLURGICAL CHEMISTRY.

designed to attain the greatest accuracy and rapidity in analysis, in order that the skill in manipulation acquired in the technical school may be equal to that demanded of the practical analyst.

In the equipment of a laboratory for metallurgical chemistry in the Department of Mining and Metallurgy of Harvard University, in the year 1901, on the principles given above, a few innovations, in arrangement and appliances, were introduced, and, in consequence of the resultant advantages, they are taken as the basis of the present paper.

The general arrangement of the working-tables, shown in Figs. 1 and 2, is similar to that usually found in chemical laboratories, the unusual feature being the distribution of the hoods on the tables, in such a manner that each student has an individual hood-space at one side of his desk. In metallurgical analysis, a large part of the work of most determinations must be done under the hood; and evaporations, in order to be car-

FIG. 2.



DETAILS OF EQUIPMENT OF A WORKING-TABLE.

ried on with due rapidity and safety, should be constantly under the eye of the operator. This arrangement of hoods and their construction of glass enable the student to carry on simultaneously evaporation, or other hood-work not requiring constant manipulation, and such operations as filtration, washing of precipitates, titration, etc. Each desk is also provided with a sink, gas, compressed air and suction, so that there are few opera-

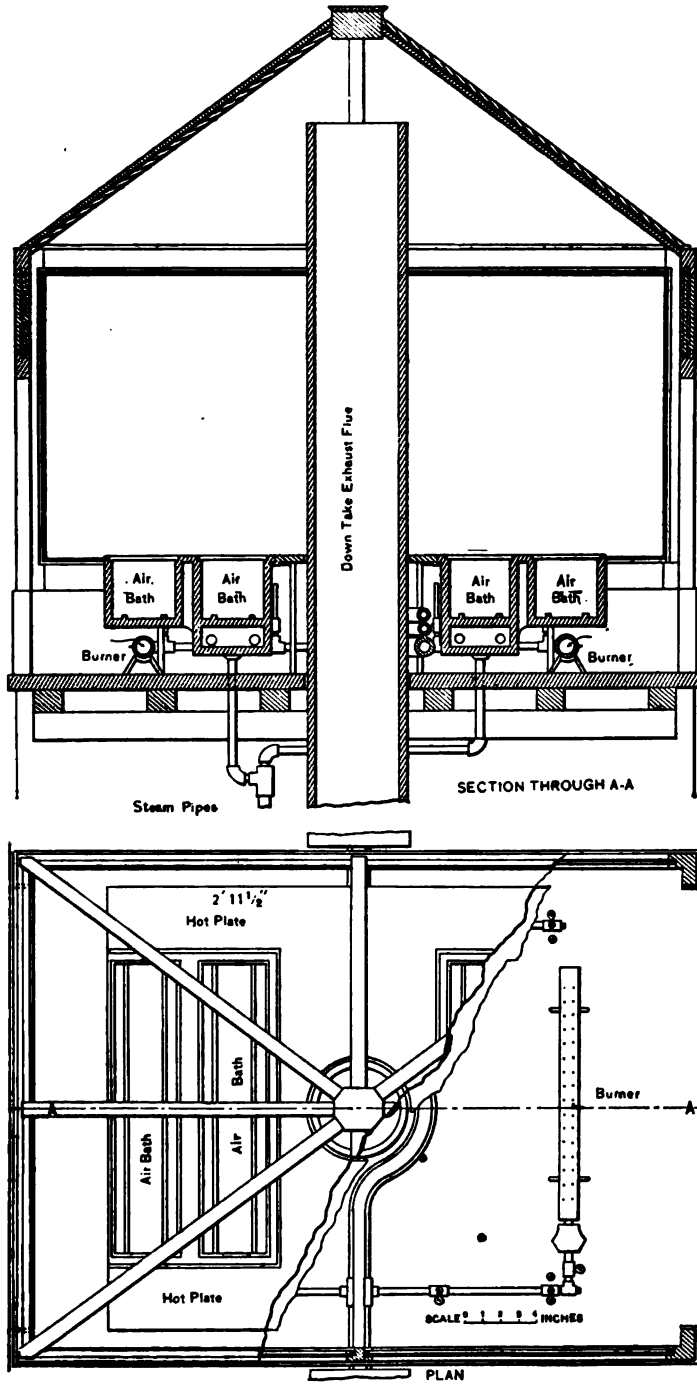
tions except weighing that cannot be carried through to completion without the necessity of leaving the desk.

Besides the advantages already noted in favor of the use of glass for the hoods there are the additional advantages that it is attacked by only a few chemicals, is easily cleaned, and does not obstruct the light. The sides of the hood should be supported on a base of the same material as the floor, so that the glass may not be brought too near the burners under the hot plates. The floor should be made sufficiently strong to support a moderately heavy weight. It should be constructed of non-combustible material, that will resist the action of chemicals and will not crack on heating. Slate has all the necessary qualities except that it is cracked by heat, and soapstone lacks the required strength. Portland cement reinforced, and provided with a tile-surface, would make perhaps the most satisfactory floor for a hood.

By an exhaust fan, the fumes are drawn through a pipe of glazed drain-tile, from near the top of the hood, down through the floor, and discharged from the building through a large main. The exhaust is sufficiently strong to dispense with the necessity for doors to the hoods.

The details of the equipment within the hood, shown in Figs. 3 and 4, consist of cast-iron hot plates, air baths, and hose-nipples for the attachment of Bunsen or other burners as required. The iron hot-plate has the form of the letter I, the open ends being filled by the air baths, the whole making a rectangular heating surface, having dimensions roughly proportional to those of the bottom of the hood, but somewhat smaller. The two pieces in which the hot plate is cast are held firmly together, by bolts, about the exhaust pipe in the center of the hood. The plate stands on iron legs, and in turn supports the air-baths on three sides by a flange. The two air-baths for one side of the hood are cast in one piece, as shown in section. This section also shows that the adjacent walls of the baths are cast 0.75 in. apart for the insertion of an asbestos board between them, which extends to the floor of the hood and prevents the heat from passing too freely from one bath to the other, and admitting of the easy control of the temperature in each bath independent of the other.

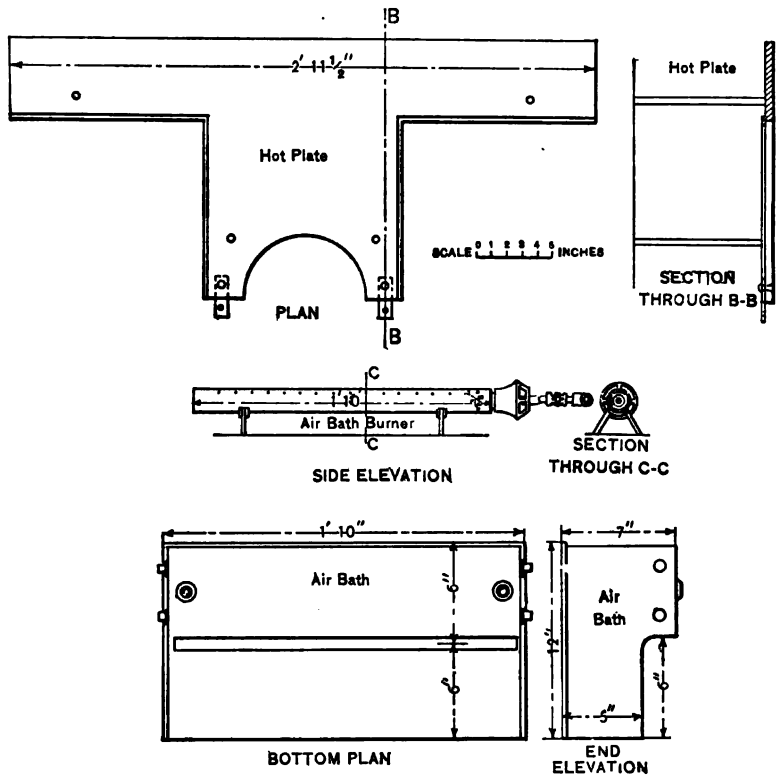
FIG. 3.



DETAILS OF EQUIPMENT OF HOOD.

The burner by which the front bath is heated, shown in detail in Fig. 4, consists of an iron-pipe, or barrel, an air-mixer, and a neck connecting the two. The burner, which rests horizontally, has two rows of small perforations for the emission of the mixed air and gas. The gas enters the mixer through a small jet and draws the air into the burner with it. The air in the rear bath is kept at a constant temperature by steam deliv-

FIG. 4.



DETAILS OF HOT-PLATE, BURNER AND AIR-BATH.

ered from the fire-room boiler to the steam-chest beneath the bath. The steam-chest is cast with the baths, and is connected by pipes with the boiler after the manner of a radiator. The heat is regulated by a steam-valve, the temperature in the bath depending upon the pressure in the boiler and the amount of steam admitted to the steam-chest. The baths having, therefore, only dry air, may be used for drying-ovens as well as for the other purposes to which steam and air baths are adapted.

All iron apparatus in the hoods is protected from the attack of chemicals by an occasional application of graphite in the form of stove polish.

Covers and rings for the baths are best made of porcelain, though strong asbestos board is a very satisfactory substitute. Burners for the hot plates may be such as are described above for the baths, or short Bunsen burners attached directly to the gas-pipe may be used. A Bunsen burner with tube less than half the usual length may be used without danger of snapping back, if the air is admitted through small openings to all sides of the gas jet.

There are two hose-nipples for compressed air and two for suction at each table. The compressed air and partial vacuum are each maintained at about 5 lb. per sq. in. by a small steam-pump in the fire-room. In addition to the usual apparatus used in connection with suction and compressed air to facilitate chemical work, there have been added mechanical stirrers of a very simple pattern operated by compressed air. The stirrer consists of a metal disc with small cups fixed to the periphery, after the manner of an anemometer, to which the jet of air is applied. The disc, fitted with a glass-rod spindle, is set up to rotate in a horizontal plane.

On the tables at each end of the laboratory are appliances for general use, including a condenser for distilled water, combustion furnaces, poles for electro-chemistry, a mechanically operated agate-motor for grinding samples, and an agitator for chlorination and cyanide tests of gold-ores.

This laboratory at Harvard University has been used for two years, with very gratifying results. Not only has the quantity of work in the several courses been much increased, but the student's interest in the work is also better maintained by the improved facilities for carrying through his determinations with despatch, thereby improving the quality of the work and enhancing its educational value.

A Decade in American Blast-Furnace Practice.

BY F. LOUIS GRAMMER, BALTIMORE, MD.

(Atlantic City Meeting, February, 1904.)

THE iron industry has been so markedly the cynosure of all eyes, that a sense of weariness has overtaken many on-lookers, and a new wonder is desired.

While the commercial phase of the iron industry has necessarily engrossed the public attention and probably is the more worthy of record, still the mechanical and metallurgical phases have recorded conspicuous advances and merit attention.

In order to appreciate the present condition of blast-furnace practice, it is necessary before enumerating the advances made during the past decade to give a brief discussion of the tendencies in the methods of administration as well as the conditions governing supremacy, for these factors have modified the aims of the manager.

It might also be well to apologize for a record of the advances made during such a short period as a decade, were it not that, for the individual, time is marked as much by the impressions recorded on the brain as by the jumps of the second hand of a watch. Furthermore, this article is written in order to carry out a project of the late Dr. Egleston, to have a record made each decade. I regret that the data collected by him, and forwarded to me, end with 1896, and therefore do not include the interesting results which followed the introduction of Mesabi ores. The perusal of the notes is melancholy reading, somewhat like the feelings we experience in going through a graveyard. It is a record made up of so many plants, which have ceased to be factors in the iron world, if they are not entirely moribund. Historically, they would be interesting, emphasizing as they do the abnormal and somewhat overestimated importance of furnace lines, taken without proper reference to volume of blast and nature of stock used. I regret, however, that the time at my disposal does not permit of this presentation.

Nowadays, the administration at some plants is such that the superintendent has become a train-dispatcher or a burden-clerk; and this is particularly the case in those plants in which the engineering duties have been divorced from the executive functions of the superintendent, and where the preparation of ores is made at the mine rather than at the plant. At such plants, usually consisting of four or more units, the proper feeding of the furnaces with a mixture, which the past has demonstrated will not make it ill, has become one of the superintendent's primary duties. The traffic management also is important and at several plants, running from 8,000 to 14,000 tons of raw material daily, excluding metal made, is equal in tonnage to that of many railroads. In addition to the traffic consideration, the cost and supplying of labor in these troublesome labor times, is properly deemed ample responsibility for the superintendent.

This distribution of duties is in line with the general tendency of the age to differentiate the duties of the laborer and the under executive, as well as to integrate between wider limits the responsibilities of the fewer and more powerful higher executives, and has resulted in many benefits and some colossal errors. It has required the superintendent to be more a reader of events and men, while the engineer becomes more concerned with new devices to harness nature.

In the most powerful companies, this division of duties has usually resulted in good, but in those plants, like Peter Pindar's razors, "made to sell," many mistakes have been made.

We can look around and see plants well arranged to make iron, but no iron mines to supply them. Others have fine mills, but no furnaces; others, no market. And so the whole decalogue of managerial sins, resulting from a bureaucracy or directorate of untrained iron men, can be run.

While largely due to perfected means of communication, the present state of affairs is indirectly traceable to the Bessemer process, whose Gargantuan appetite suggested the assembling of a large number of units near together. It affords a picture presenting marked contrasts to the time of Baron Stigel, whose return home, after an absence, was heralded by bonfires from the hilltops and the booming of cannon. It was a paternal management and included such diverse industries as charcoal-burning, road-building, farm-management, pottery-making

and forge-running; a Pooh-Bah list of duties, picturesque, if not conducive to the rapid development of an industry! His contract that a rose a year should be deposited on his grave forever, at a church in Manheim, Pa., in lieu of a ground rent, is not one that we imagine the iron masters of to-day would make.

The present conditions are even very different from the management under the Coleman family, so long a powerful and beneficent influence in the industrial world. Their manager was regarded as a family retainer with interests permanently indentified with the family welfare. Under their employ the superintendents were as much engineers as managers.

The Bessemer process, by inaugurating the general policy that each process is the servant of the succeeding process, made the furnace the servant of the converter, and the converter of the rail-mill; and in the other direction the coke-oven and the ore-mine bore the burden of the furnace superintendent's criticisms. A marked instance of this is told by a Mr. C., who ran the furnaces for a large plant in the middle West. At one time, he was being hauled over the coals rather roughly by the owner, because the mill was not getting the silicons it wanted. At last in petulance he said, "Are there no rights of the furnaces that the mill is bound to respect?" The answer received was final, "Not one."

Such a condition of affairs made the furnace superintendent's prestige depart, and his responsibilities are aptly described by one manager as being limited to the coke-barrow and the tapping-hole. On the other hand, the mechanical engineer and accountant came to the front.

As a result of those changed conditions we now find casting-machines introduced at plants where the highest mechanical efficiency is represented by a clever use of wedge, sledge and crowbar. We find automatic chargers used where the double-ring bell used so successfully by Mr. Firmstone and others should be used. We find the 100-ft. furnace treating magnetic ores and having a number of moves, which as managers and metallurgists we were taught to deplore. In short, if the advances became more rapid they were not always judiciously balanced. The pressure of fashion is so great that many do not consider whether it is personally becoming; in fact, local conditions are ignored.

On the strategic side we find, that, whereas formerly the ownership of a good ore-property and a good location near the market was supposed to be all that was necessary for independence in manufacturing, now one should own the cars, coke-ovens, coal-mines and frequently the railroad and the stone-quarries, as well as the plant of his former purchaser.

Therefore, at the plant not independent of fuel-supply, or whose fuel is expensive, fuel-economy is the first consideration, while at the plant in a disturbed labor market, labor-saving devices are of paramount importance, and at the plant which controls the market, the size of the output is the first virtue. Obviously, with such a variety of aims the chief requirement of one plant may be of secondary importance at another.

In looking over the development of furnace-practice, four steps or incidents appear as the more important factors: (1) The use of waste-gas under boilers; (2) the heating of the blast; (3) the use of coke as a fuel; and (4) the use of Lake ores. Each of these steps has resulted in a doubling and trebling of the output which was possible before their introduction.

Of course, improved refractory materials and better engines were essential, as was also a knowledge of chemistry, but these influences should be regarded as secondary and logical sequences to the others. The better application of the knowledge classified under these four heads represents the development in America.

The earlier volumes of our *Transactions* are replete with papers concerning the analysis and fusibility of slags, scaffolds, frozen furnaces, titaniferous ores, dirty walls, sulphur in metal and a host of troubles resulting from a scattered and imperfect knowledge and an uncontrolled condition. As the facts became clearly known, men avoided trouble on the principle that "prevention is better than cure."

A decade ago lines of furnaces and cooling devices occupied the thoughts of the furnace world. Since then the advances made may be classified under headings as follows:

(1) Conveyors and other mechanical improvements.

(2) Metallurgical by-products.

(3) Miscellaneous.

Exigency of space precludes all but a simple mention of many of the steps, as a division of duties has been attended by a flood

of ideas developed along the avenues where the intrusive finger of modern accounting has shown that leaks existed and improvements could be made.

Mechanical Conveyors.—The use of Lake ores increased tonnage so rapidly that the simple handling of ore, coke and stone became a serious problem, and, as the properties became more and more under one control, the unnecessary moves of ore into boats to be unloaded on to docks, and to be again reloaded, were reduced. Steam-shovels, like the Marion, Bucyrus and the Thew, were used to dig the ore out of the mines and to transfer it from the stock pile into the car. Messrs. Hulett, McMyler, Hoover & Mason, and the Brown Conveying Co., devised means of economically unloading the ore from vessels of many hatches. By means of these devices, and large piers at the shipping points, the hours when the boats were idle were minimized.

The rolling-mill principle of keeping the passes of the rolls full of metal, and the transportation-virtue of keeping as many trains to the mile as is compatible with safety, was applied to the movement of the raw materials from the mines to the furnace. The steel cars reduce the cost of unloading by their steep bottoms, and these, with the bin-system, as at Duquesne, and car over-turners and bridges, as at Youngstown, represent the chief changes in the matter of handling raw material.

Mr. Axel Sahlin, in our *Transactions*, has comprehensively sketched the general points of mechanical transportation in detail, and it is therefore not necessary to enlarge upon them in this paper.

In moderate climates, where labor is expensive or troublesome, these devices have frequently paid well, and are sometimes necessary where the season for bringing the ore from mine to furnace is limited. Personally, I think that the skip-hoist, the last labor-saving step before the raw material enters the furnace, has not infrequently been introduced where the double-ring bell would have been better. This restriction, however, is not applicable to the general run of Lake ores.

Messrs. Walter Kennedy, M. A. Neeland and E. G. Rust designed skip-hoists which may be regarded as representing the three types. Mr. Walter Kennedy's descending-bucket passes at the side of the ascending-bucket, and acts to some extent as

a counter balance; in the hoist of Mr. Neeland, one bucket only is used, and in that designed by Mr. Rust, the ascending- and descending-buckets pass over and under each other.

Use of Wash Ores.—Mr. Firmstone showed me some charts of silicon and sulphur in basic metal made from wash-ores varying from 10 to 12 per cent. of silica, which in their regularity and percentage of off-cast compared very favorably with the best practice in Pittsburg; the furnace using these ores was equipped with a double-ring bell. I believe that such regularity with material of this character will astonish most of the Pittsburg iron-makers.

Valves.—In the question of valves, some attempt has been made to improve on the Mushroom and Berg valve-seat, the Spearman and Kennedy burner, and the cold-blast valve; save that they are made somewhat thicker and larger, they are substantially as they were a decade ago. The cutting action of Mesabi ore has suggested the multiplication of false seats and flanges on the stoves.

Tap-Hole Gun.—The Vaughn gun makes the work of stopping the hole easier on the men and is especially satisfactory if operated by compressed air.

Direct Process.—The extensive adoption of that very important link in the iron-plant—the direct process and the mixer—has suggested the undesirability of carrying a gang of specialized workmen all the week simply to carry out and break the iron on Sunday, when the converter is idle. This condition of practice has resulted first in cast-iron chills in the cast-house and then in the invention of casting-machines.

Casting Machines.—Mr. James Scott and Mr. Uehling have conjointly perfected the Uehling casting-machine, which is a monument to their perseverance in overcoming many obstacles. The Heyl & Paterson conveyor uses lamp-black in place of lime, and pressed-steel pans in place of cast-iron ones. I have always found the electric breaker at Duquesne, if used in connection with the iron chills, less expensive and more satisfactory, though the iron cast in it is not so easily handled nor so attractive in appearance. Pig-iron casting-machines require more attention and are more easily thrown out of order than the electric breaker. Besides the above mentioned there are the Davies & Aiken pig-iron casting-machines, whose merits commend them

to some. The iron-chills were necessary because of the rapid growth of the basic process, in which sand and silica are most objectionable materials. In addition, the use of iron-chills saves the labor of molding the pig-bed in sand, and from them the idea was extended to making the runners and the skimmer of iron; the latter to a great extent doing away with boils.

Slag-Car.—The Weimer slag-cars, lined with iron thimbles, have been introduced practically to the exclusion of all others.

Slag Disposal.—In a few plants the slag is run into pits and granulated, and then lifted on to cars by means of cranes and orange-peel buckets. This method is very economical and is especially applicable when it is necessary to dump the cinder at a great distance from the plant, more especially if it can be utilized for manufacture into cement. One disadvantage of granulating the slag is that it may cause annoyance by creating a cloud of steam at the time of casting the iron. In some plants the slag is run into dishes, or metal pans, working on a conveyer, and after it cools is broken up and carried away for use as railroad-ballast, a method which is advisable if the slag is non-slacking.

Recording-Gauges, etc.—The introduction of more recording-gauges for steam, air-blast, vacuum-pressure, and temperatures of blast and escaping furnace-gases, marks the advance of furnace practice into a state of better control.

The extensive introduction of the direct process and the pig casting-machine has, in a few cases, caused the abandonment of the cast-house, except a small building which is required to cover the cinder and the iron in their passage to the ladles.

Dust-Pockets.—The use of Mesabi ores has made an increase in the number of dust-pockets on the gas-main. These pockets as well as the dust-catcher are now suspended above a track, so that when the pockets are dumped the dust will fall into the cars which have been run underneath them.

Ladle Drying.—The iron-ladles, in the absence of natural gas, are frequently dried out by burning the blast-furnace gas brought to them in a small flue.

Gas-Flues.—In a few American plants the gas-flues are not lined with brick, but the great majority use cheap fire-brick and do not copy foreigners in this practice.

At some places, as at Braddock, a satisfactory water-seal valve is used to isolate a furnace from others on a common system

of gas-flues, for which purpose the Rothoff valve is also extensively used.

Gas-Mains.—Almost all gas-mains are now overhead, and underground mains are avoided. Not only is the steam system universal in the modern plant, but the tendency is to make the gas-flues universal. If one of the furnaces is cold and its gas, therefore, not suited for stoves, the gas from the three other furnaces in the system will help to maintain the stove heat on the one that needs it; in a similar manner the gases from the many furnaces tend to keep the steam-pressure regular.

Boilers.—The introduction of water-tube boilers marks the further attempt to obtain the full power-possibility of the escaping gas; the Babcock & Wilcox, the Sterling, the Cahall being those most generally adopted. The numerous water-softeners, generally based on the principle of having the carbonates precipitated by lime and the sulphates decomposed by soda, have enabled water-tube boilers to be more extensively introduced. The old idea that because there is an excess of gas and because of the ease with which scale can be removed, cylindrical boilers should be used, has passed away.

Use of Compressed Air.—Compressed air in place of steam is being quite extensively used for such purposes as the operation of the furnace-bell and the mud-gun. It does away with the danger of burning the men and of having the water condensed around the tapping-hole and freezing around the furnace-top.

Steam-Pressure.—The economies in the use of blast-furnace gas have become so extended that we now economize also on the steam obtained from the gas. In place of the 80-lb. pressure of 12 years ago, and the 60-lb. pressure of a decade earlier, we now use from 120- to 150-lb. steam-pressure in connection with compound engines, condensers and feed-water heaters. The engines are of heavy frame and the air-valves are positive acting, which gives a higher efficiency of delivery.

Without the modern engine-equipment it is probable that the present phenomenal outputs could not have been attained. Despite the fact that a finely divided ore is reduced more rapidly than a lumpy coarse ore, its use requires a greater blast-pressure, and a larger volume of blast, else its fineness will not be taken advantage of; therefore, if the increased pressure

and volume could not be supplied, the output would be smaller than when using lump ores.

I remember visiting a plant where this fact was not appreciated by the owner. It had too many engines and too many boilers, but the furnace could not get a sufficient quantity of blast to satisfy its hearth area. The steam-pressure was only 60 lb. per sq. in. and the blast-pressure only twelve.

The Southwark Foundry & Machine Co., the E. P. Allis Co., the shops of the Tod Engine Co., and, latterly, several others have met the new conditions. Not, however, without some tribulation, for a 100-ft. furnace requires more work to be done than a 90-ft. one. This latter dimension is, in my opinion, metallurgically more desirable.

The hoist-engines in the majority of cases remain extravagant users of steam. Several plants have introduced electricity to operate the hoist.

In a few plants happily designed, where the height of the furnace is not over 90 ft., there is a surplus of steam obtainable from the furnace-gas after supplying the power-demands and the stove-demands. These plants sell the surplus steam to the adjacent mill. In many other plants the increased blast-pressure resulting from the excessive height of the furnaces, or other conditions, have nullified the steam-economy, resulting from the improved machinery.

In some instances it has been deemed advisable to use the surplus gas to attain a higher temperature of blast (with the view of having a lower fuel-consumption), rather than to sell it to the mill.

Hot-Blast Stoves.—In hot-blast stoves, the bottom-rings are now made of such a height that the riveting on of door-frames, ports and branches is done without crossing a seam. These plates are also made very much heavier than formerly because of the higher blast-pressure now used. Stoves of the central combustion-chamber type seem to be gaining in popularity, and brick specially shaped for the checker-work continues to be extensively used.

Refractory Brick.—The brick-manufacturer has been fully abreast with the requirements, and supplies a cheap brick for the ladle-lining, and a brick free from iron suitable to withstand abrasion for the furnace-top, and one free from alkalies

and bases for use with high heats. Prior to the year 1890, a blast-furnace campaign lasted from 18 to 30 months, now it exceeds eight years, and several furnaces have each produced more than a million tons of pig-iron with one lining, which has reduced the relining-charges per ton of iron produced, from 50 cents to less than 15 cents.

Shields.—A number of shields and protectors have been devised to protect the lining of the upper part of the furnaces from the abrasions of the stock rolling off the bell against the top walls. Of these that I have seen, the best is a suspended sheet of heavy rolled steel, which was introduced by Mr. Firmstone at one of his plants. As used by him, an annular opening extending completely around the furnace-top was obtained for a gas-outlet, which is better than taking the gas off at one point.

Water-Cooling.—At many plants having a small supply of water, a wooden waterfall is used for the purpose of cooling the condensing water for repeated use.

By-Products.—There has been a rapid growth in the use of slag-cement which is placed on the market under the name of "Puzzolini," a name derived from the natural cement rock of Italy. Slag-cement is used as a substitute for Rosendale cement for purposes not requiring the highest degree of reliability. Among other places, it is manufactured extensively in Chicago, Ill.; Youngstown, Ohio; and Sparrow's Point, Md. The process of manufacturing slag-cement has been frequently described, and is being so improved that the consumption of this product will probably increase.

In charcoal-manufacture, acetate of lime and wood alcohol are obtained as valuable by-products, and the by-product coke-ovens yield ammonia, tar and gas. Each of these subjects is worthy of an individual monograph, especially the by-product coke-oven, with its promised economies in fixed carbon, yield and labor.

Flue-Dirt.—The loss of flue-dirt in the treatment of mixtures containing a high percentage of Mesabi ore, has suggested the use of gas-washers and briquetting-machines. The Steece and the Roberts washers are those in most general use. The Henry S. Mould briquetting-machine has been introduced at several plants to recover the ore that has been blown over from the furnace. The loss through flue-dirt can unquestionably be les-

sened by a study of the conditions outlined in my recent paper in our *Transactions*, "Flue-Dirt and Top-Pressure in Iron Blast-Furnaces."[1]

Saving of Gas.—The introduction of a double bell, preventing the escape of gas during the lowering of the charge, has resulted in a saving of from 10 to 15 per cent. of the gas. At several plants using a single bell, the average time during which the bell was open exceeded two hours and forty minutes in 24 hours.

Blast-Furnace Working.—Impressive as is the metallurgical practice in America, it exhibits inventive ability less than natural resources. We owe more to the regions named after that emissary of peace, Pere Marquette, and the tribes he went out to civilize and Christianize (Menominee and Gogebic), than we do to original research. We have, it is true, the Uehling pyrometer of American origin, which is an instrument of great precision and of great value to the furnace-man. Our records, however, are characterized by bold application rather than new ideas.

Our high furnaces do not reflect great credit on their designers, though in justice it should be said that most furnacemen were not in favor of 100-ft. heights.

I have personally inspected more than sixty furnaces; and I find that the fuel-consumption, other conditions being equal, is lower on furnaces of from 70 to 80 ft. in height than on furnaces exceeding 90 ft. While Dr. Egleston's records do not include any very high or very large furnaces, the best fuel-consumptions he gives are from furnaces in the neighborhood of 75 ft. in height.

With very irregular ore or fuel, and very expensive coke, it is a question whether a very large output per furnace is desirable. A small bad cast is more easily taken care of by the mixer than a very large one. The principal reasons, however, why our fuel-economy has not improved (in fact, it has gone backward) are as follows: The coke-ovens have been insufficient to meet the increased demand; and in order to increase production, the time of coking has been shortened, resulting in a poorer quality of fuel. Then again, the shortage of cars has caused many furnaces to be repeatedly banked, which has con-

[1] *Trans.*, xxxiv., 92.

sequently increased the coke-consumption. Many cokes formerly considered too high in ash, and therefore low in carbon, have been put on the market. Finally, the furnace-mixtures used have been leaner.

The stove heating-capacity has not kept pace with the blowing-power; hence lower temperatures of blast have been used, resulting in higher fuel-consumption. In several instances it was considered desirable, in view of a brisk market and large profits, to use the furnace-gas for making more blast, rather than to save the coke, by using a higher temperature of blast.

Generally speaking, the silicon-requirements for Bessemer iron have been lower, depending upon the location of the plant and other conditions. The average percentage of silicon may be taken at 1.1 per cent. in summer and 0.9 per cent. in winter. The lowering of the silicon-demand has been in the furnaceman's favor. A brisk market also has lessened the severity of the sulphur demands of the mill. I think that in some quarters there is now greater tolerance of sulphur than 10 years ago. In other quarters the metal must be remelted if it contains more than 0.05 per cent. of sulphur.

Owing to the improved preparation of raw material, the furnaceman is supposed to be able to keep the sulphur down without the use of manganese; and as high manganese-percentages, through the great spluttering they occasion, prevent the forcing of the work in the converter, this element is now a greater detriment to iron-ores than it was in the beginning of the past decade.

The increased purchase of ore containing a higher phosphorus-content has accompanied the rapid extension of the basic open-hearth steel-process. It is not unusual, upon the shutting-down of the converter on Saturday afternoon, while the metal is being run into the chills or through the pig-casting machine, to put a basic mixture in the furnace: a procedure which is especially desirable if the furnace-capacity of direct-process metal is sufficient for the converter-capacity and need not be supplemented by remelting the Sunday's product in a cupola-furnace. The direct process, with the great advantages afforded by the closing-down of the cupolas, has been greatly extended; and molten metal in Pittsburg and Cleveland is carried in 20-ton cars for a distance of more than five miles.

The use of multiple tuyeres has not always been attended with satisfaction; and a more conservative estimate of their benefits now prevails.

The drying of furnaces preparatory to "blowing-in" now occupies much less time than was the practice a decade ago. Many think a week is quite sufficient. During the blowing-in period the burden is now increased more rapidly, and the quantity of wood used is very much smaller than formerly. I know a very successful operator who uses no more than a cart-load of wood for blowing-in a large furnace, and a few who light the furnaces in starting by means of red-hot iron-bars introduced through the tuyeres, while the blast is on. Red-hot charcoal also is satisfactorily used by some, who blow it through the tuyeres during the starting of the blowing-in. The hot-blast stoves can now be heated higher previous to starting than was formerly practicable, owing to the use of natural gas or the universal gas-main.

A saner treatment of the tap-hole now prevails, due to the recognition that an exceptionally large product for a single day means little; and, as a consequence, the last portion of iron in the crucible is not drawn out by prolonged blowing at the tap-hole. The blowing at the tap-hole leads to break-outs, on account of the heating up of the furnace-front.

The blowing away of furnace bell-and-hopper by slips, which frequently occurred after the introduction of Mesabi ore, is now unusual. This generally disastrous irregularity was then attributed to a so-called dust-explosion; but I think this assumption is wrong, and that irregular working, even with high percentages of Mesabi ore in the charge, can be obviated.

The proportion of Mesabi ore used in the ore-mixture has, in exactly a decade, increased from 25 to 100 per cent., a furnace in Pittsburg having been blown-in recently with the ore-mixture composed entirely of Mesabi ore.

While the ingenuity of American metallurgists savors less of the lamp than that of European engineers, still in the recognition that fine ores are quickly reduced in the furnace (analogous to the manner in which salt enters more readily into a solution if the more finely divided, and its corollary, that a finely divided material must be given a large volume of solvent in order to dissolve it rapidly), and in the application of this principle, American engineers have acted with great promptness and have received enormous returns.

The introduction of gas-engines at the Buffalo plant of the Lackawanna Steel Co. marked an important epoch in blast-furnace practice. According to Mr. H. G. Scott,[2] by eliminating boilers, and thus combining the duties of boiler and blowing-engine, economies are promised amounting to 20 per cent. Mr. Scott, who is engineer to Cochrane & Co., at the meeting of the Cleveland Institute of Engineers, December, 1902, said that the economies resulting from the introduction of gas-engines vary with the plant, being influenced by stoves, mixture, etc. Cochrane & Co. obtained an efficiency of 25.43 per cent. with gas-engines, as compared with 6.732 per cent. for boilers and engines combined.

Through the courtesy of Mr. Wehrum, formerly the general manager of the Lackawanna Steel Co., the following data on the company's gas-engines have been furnished:

"The distribution of the units of horse-power of blast-furnace gas-engines installed in Europe prior to February, 1902, was:—England, 600; Italy, 1,800; Russia, 2,230; Austria, 2,840; France, 7,400; Belgium, 7,600; Luxemburg, 15,400; Germany, 44,665; making a total of 82,535 h.p.

"Blast-furnace gas in gas-motors is 37 per cent. higher in efficiency than that used to produce steam, and I estimate that the engines introduced at the plant of the Lackawanna Steel Co. under my administration shows an economy of fully 300 per cent. more than that of the single-condensing steam blowing-engines, which is equivalent to a saving of $12.50 per horse-power per year, by the introduction of blast-furnace gas-motors. Gas-engines are now installed and in operation at the Buffalo plant of the company to the extent of some 5,100 horse-power."

The above-mentioned quantity of horse-power (82,535) has since been increased to 297,050, which is distributed among the various gas-engine makers as shown in Table I.[3]

TABLE I.—*Blast-Furnace Gas-Engines (Exceeding 200 H.P.) Completed or in Course of Construction Oct. 31, 1903.*

Name of Maker.	Deutz.	Koerting.	Nuernberg.	Oechelhaeuser.	Cockerill.	Total.
Number of engines...	123	70	57	41	116	407
Total horse-power....	49,225	83,475	61,350	27,400	75,600	297,050
Average horse-power	400	1,192	1,076	667	652	730

[2] *London Engineering*, May 22, 1903.

[3] *Iron Age*, Jan. 14, 1904.

For those who wish to study this subject further, I refer them to the interesting brochure published by Mr. Wehrum.[4] Considering the fact that about two-thirds of the gases are now used for the production of power and one-third for heating the stoves, the subject is well worthy of study.

Mr. Uehling[5] makes a statement, that for each ton of pig-iron produced per hour there will be available 800 h.p. for sale or for use in connection with the rolling-mills connected with the blast-furnace plant, and Mr. F. duP. Thomson, who assisted Mr. Wehrum at the Lackawanna Steel Co.'s plant, is of the opinion that 500 h.p. per ton of pig-iron produced per hour would more nearly approach practical working.

The blast-furnace has always been regarded as representing a high degree of efficiency. In the direct process, the heat contained in the molten iron has been saved, and doubtless ere long the heat of the molten slag also will be utilized. But it is in the line of using waste-gas that our signal economies have been scored. First, in using it under the boilers, then, in using it under the stoves, then, in sealing the top of the blast-furnace with a double bell, then, in selling the excess of waste-gas to the mill in the form of steam, and now, after continuously demanding more from it, we hope to receive more by the introduction of gas-engines.

In the Metallurgical Congress, which I suppose will form a part of the Louisiana Purchase Centenary, it will be appropriate to record the progress of that plant of unrivaled natural resources, viz., the Colorado Fuel and Iron Company, situated in what was then the great American Desert, and what is now one of the greatest coal-fields and ore-deposits of the world.

Although our coke-consumption remains between 1,750 and 2,100 lb. per ton of metal produced, our daily output per furnace has jumped from 350 to 500 tons and more in a decade, and the total yearly output of pig-iron, according to Mr. Swank, has grown from about 8,000,000 tons in 1893 to 18,009,252 tons in 1903. This rapid increase in the production can be understood when it is known that one 500-ton furnace has been erected and has produced iron within one year and one day after the pick was first driven into the ground.

In the iron world it has been proven that high wages need

[4] *Studies of Blast-Furnace Gas and Its Most Economical Use*, Scranton, 1900.

[5] *Stevens Institute Indicator*, January, 1903.

not mean increased cost of production. The region between the Great Lakes and the Connellsville coal-field is still regarded as being a section where the most advantages are found, possessing easy access to high-grade cokes, the best ores, the most skilled and ambitious labor, the greatest mechanical ingenuity and the best markets. Colorado and Alabama also hold strong positions.

While the decade has recorded the abandonment of many plants economically unfit, it has been a period of great activity in building new plants in the localities above mentioned, as well as in Canada and Mexico. The tendency has been towards fewer units and larger units, towards a keener appreciation of the reduction in cost resulting from using a large output as a divisor, particularly with reference to the reduction of the cost of management and fixed charges.

A series of observations by barometer and hygrometer emphasize the disadvantages of high humidity, and have led in a few instances where there were hot, moist engine-rooms to the supply of air from outside the building by means of especially constructed pipes. This also is a line of investigation receiving the attention of some of our foremost furnace managers.

I fully realize that many names entitled to special mention for having taken a prominent part in the astounding progress of our industry, have been omitted from this paper; and I can only trust that no one will feel that a discourtesy has been intended. I cannot, however, close this brief record without the mention of the one whom I think has most clearly discerned the trend of events.—Mr. James Gayley early saw the possibilities of Mesabi ores and the methods of handling them, as he also saw the advantages of a low bosh and a large hearth. Both in justice and with a sense of personal loyalty, I feel that the name of Mr. James Gayley should be placed permanently among those who have contributed so much to the splendid development of the blast-furnace practice during recent years.

A worthy review of the blast-furnace advance cannot be compressed into the short limits of this article, nor had I sufficient time to treat it other than I have done. This paper has been written in the hope that it might awaken the members to an appreciation of the scarcity of papers on blast-furnace practice that have been published during the past decade. Our *Transactions* were formerly replete with such records.

The Use of High Percentages of Mesabi Iron-Ores in Coke Blast-Furnace Practice.

BY W. A. BARROWS, JR., SHARPSVILLE, PA.

(Atlantic City Meeting, February, 1904.)

MESABI ores differ from all other soft hematite-ores of the Lake Superior region in having but little or no binder. When exposed to heat, instead of baking into lumpy masses as do the Gogebic, Menominee or Marquette fine ores, they are changed to a sandy powder; and the finer particles, having little or no tendency to adhere together in the furnace, are carried by the ascending gas-currents into the down-comers and gas-mains forming flue-dust of a fine gritty nature. The term "finer particles" is used in this paper to mean ore fine enough to pass through a sieve with 80 meshes to the linear inch. Ore coarser than this is not likely to be carried over except in case of a slip, or an explosion in the furnace.

The higher the percentage of fine particles in an ore from this range, therefore, the greater the amount of dust likely to result from its use. This statement, however, does not hold good in case of fine ores from the other ranges, and ores from the older ranges, carrying as high as 20 per cent. of material fine enough to pass through a 100-mesh sieve, have been in successful use for years, and have left no records of their causing trouble in furnace-practice by dust, slips, or explosions, so common since the introduction of Mesabi ores.

Mesabi ores containing less than 12 per cent. of material which will pass through an 80-mesh sieve are considered of good structure; from 12 to 18 per cent., fair structure; and more than 18 per cent., poor structure.

It was at first supposed that all these fine particles were carried over when the charge was lowered into the furnace, or during a slip, but the presence of partly reduced iron oxides in the flue-dust, even when no slip has occurred, proves that a part of the dust at least must have been low enough in the

furnace to have been acted on by the reducing gases and subsequently carried up and over with the escaping gas-currents.

Notwithstanding the difficulties experienced in working Mesabi ores, the facts that they can be so cheaply mined and that ore of good chemical composition exists on this range in immense deposits, have forced their use. The rapid depletion of the Old Range reserves, particularly of ores of Bessemer grade, has rendered it absolutely necessary from an economic standpoint that the maximum percentage consistent with safe furnace-operations should be taken from this range, and that every energy on the part of the furnace superintendent should be devoted to adapt the furnace-practice to this end. In fact, his value to his employer in some districts is dependent on his ability to use this ore liberally in the charge. Fortunately, not all the ores from this range are of poor structure; some, in fact, contain little or no objectionable material, but most of the ore mined from the deposits lying near the surface, including some of the largest and most cheaply operated mines on the range, do contain a considerable proportion of ore in a state of very fine division. In a greater measure their cheapness recommends them and forces their use.

A number of the properties first opened up on the Mesabi range furnished ore of very poor physical structure, and much damage to furnace plants followed its use. This had become so serious about eight years ago that all ore from this range was considered bad, with the single qualification that some were worse than others. Recent operations, however, have demonstrated the unfairness of this conclusion, and furnacemen now have generally come to recognize the fact that a great many ores from the Mesabi range are of good physical structure, and that, with modified practice, they work satisfactorily in the furnace, giving little trouble from dust and causing no slips or explosions.

Operations at the Shenango furnaces, extending over a period of more than two years, have demonstrated conclusively that mixtures made up of good-structure Mesabi ores and those of the finer grades work as satisfactorily as those made up of Old Range ores and the finer Mesabi ores in the same proportions. The Old Range ores are not essential for good practice.

During the two years above mentioned, the ore mixture at the

Shenango furnaces consisted entirely of Mesabi ores and siliceous ores, as, on account of the higher freight charges paid on Mesabi ores, it did not pay to bring siliceous ore from this range; during this time more than 300,000 tons of standard Bessemer and basic iron have been made and sold, and no serious slips have resulted at either of the three Shenango furnaces. The average percentage of Mesabi ore in the charge during 1903 was 93.73, while that of siliceous ore was 6.77. The yield of pig-iron, based on the quantity sold, was 53.25 per cent.,which is practically the theoretical iron-content of the ore-mixture in the condition in which it was filled into the furnace. More than 75 per cent. of the pig-iron was chill-cast. We lost by dust and in other ways, the usual furnace-gain of from 3 to 4 per cent.

This condition of furnace-practice was arrived at only after the usual troubles with slips, explosions, etc., that have followed the introduction of Mesabi ores. Changes in the manner of filling and lowering stock into the furnace and modified lines of the top part of the furnace have done away with these troubles and, aside from the dust-problem when the mixture contains too much fine ore, our operations are as safe and satisfactory when running on all Mesabi ores as it is when running on Old Range ores.

Just prior to Jan. 1, 1901, our furnace No. 1 was operating on 50 per cent. Mesabi and 50 per cent. Old Range ores. The Mesabi ore was of good structure, in fact, I consider it the very best shipped from that range. The work then done was very unsatisfactory, slips and explosions were of almost daily occurrence, and fully one-third of the output was unsalable. On taking charge of the plant at this time I found that a change in the manner of filling and lowering the stock into furnace stopped the irregular working entirely, and since that time no serious slips or explosions have occurred at either of the three furnaces, except, when for experiment, the manner of filling was changed.

The following manner of filling has given best results:—The charge consists of 8 barrows of ore, 8 barrows of coke, 3 barrows of limestone, and 1 barrow of dolomite.

The coke-barrows at furnaces Nos. 1 and 2 each hold 600 lb., while at No. 3 they hold 800 lb. The ore-charge at furnace

No. 3 is approximately twice the weight of the coke-charge, while at furnaces Nos. 1 and 2 it runs from 5 to 7 per cent. less. The coke is filled from the bins by bulk and is not weighed. The charge is dumped at four points around the hopper and the barrow of siliceous ore, or, if none of this ore is needed, the barrow containing the coarsest Mesabi ore is dumped on the bell first. This barrow is moved around the furnace from left to right one-quarter of the circumference at each charge.

In order to understand easily the method of charging, the whole space of the hopper is assumed to be divided into four equal sections, or pockets. At first, four barrows of ore are dumped into the hopper, one to each section; then two of lime into two diagonally opposite sections; then four of coke, one to each section; then four more of ore, one to each section; then one of lime and one of dolomite in the two sections omitted in the first charge of flux; finally, four of coke, one to each section, on top of all. The entire charge is lowered into the furnace at one time. The purpose in lowering the split charge at one time is to prevent the fine ore from sticking to the side of furnace, which it is apt to do if lowered separately.

On lowering the bell the tendency is for all of the material thereon to go to the walls of furnace. The coarser parts rebound to the center, the finer parts, particularly fine wet ore, remaining where it is first placed. By sandwiching coke between the ore-layers, a portion of the coarser material is held next the walls which tends to keep them clean and to assist the charge to settle regularly. The purpose in moving the siliceous-ore or coarse-ore barrow one pocket each charge is to destroy the continuity of the gas-currents ascending through the fine ore, and to reduce the pressure incident thereto, also to prevent, in a measure, the dust from being carried over into the downcomer.

Mr. F. L. Grammer in his paper, Flue-Dirt and Top-Pressure in Iron Blast-Furnaces,[1] read before the New York meeting of our Institute, describes the condition existing in the furnace where gas is forcing its way through layers of fine sandy ore, comparing it to the bottom of a spring in which sand is kept bubbling up by the pressure of water below. In my opinion

[1] *Trans.*, xxxiv, 92.

a similar action takes place at a point much lower in furnace than is generally supposed.

It may seem that undue stress is laid on matters of filling the hopper and distributing the charge, but I believe that by charging the furnace correctly, the question of treating Mesabi ores is solved and the same results can confidently be expected as are obtained with other ores. In the treatment of Mesabi ores and those of the Old Range, I have observed no differences when changes were made in the shape of the lower part of the furnace, or in the size of the tuyeres, hearth, etc., and the changes which were beneficial in one case were likewise beneficial in the other.

The difficulties introduced by mechanical top-filling devices are many and hard to solve, and I know of no top-rigging that works satisfactorily with high percentages of Mesabi ores in the charge. The necessity of conforming to one style of filling, which allows no opportunity for experiment to determine the method best suited to the furnace-lines and the material used, is a most serious objection.

Up to the present time, the hand-filled furnaces have given better results in quantity and quality of output, fuel-consumption and uniform working,—the comparison being made between furnaces of the same size. The successful top-filling device must be elastic enough to admit of changes in filling, and in lowering the charge.

The Shenango furnaces, which are not modern in any sense of the word, are of the following dimensions:

	Furnace No. 1.	Furnace No. 2.	Furnace No. 3.
Hearth-diameter	10 ft.	10 ft. 6 in.	10 ft. 6 in.
Bosh-diameter	15 ft.	15 ft. 6 in.	17 ft.
Batter of bosh-walls to foot	3.25 in.	5 in.	3.7 in.
Batter of stock-line to foot	0.9 in.	0.9 in.	0.9 in.
Top-diameter	10 ft.	10 ft.	11 ft.
Bell-diameter	7 ft.	7 ft.	8 ft.
Height	60 ft.	60 ft.	75 ft.

Furnace No. 3 has not been operated on these lines, but is now ready to be blown in. This furnace formerly had a 13-ft. diameter top and a 9-ft. diameter bell, and made from 1.5 to

2 per cent. more dust than furnaces Nos. 1 and 2, using the same mixture. The dust, which at furnaces Nos. 1 and 2 contained about 40 per cent. of iron, contained as much as 54 per cent. of iron at furnace No. 3. This dust is recharged into the furnace, not as part of the regular burden, but in about the same quantity as it is made, so as to preserve uniform conditions. It is filled very wet and is not placed on the bell unless the furnace is ready to have the charge promptly lowered. Contrary to general claims, no bad effects are noticed from its use. Dust containing less than 50 per cent. of iron is run into the cinder-pit direct from the dust-catcher.

At furnaces Nos. 1 and 2, iron-pipe stoves are used, and the blast is heated to about 900° F.; at furnace No. 3 are four 4-pass Cowper brick-stoves built about 1880, which heat the blast, when they are fairly clean, from 1,000° to 1,050° F. The flues and the checkers are small and need frequent cleaning to prevent back-pressure on the blowing-engines.

Better results are expected from furnace No. 3 on its present lines with the narrower top, and it is hoped to keep the percentage of dust as low as that produced at furnaces Nos. 1 and 2.

The blast-pressure at the two small furnaces averages about 5 lb. at tuyeres, while at the larger furnace it is about 10 pounds.

The advantage of the narrow top with more rapidly widening walls from stock-line to bosh was noted by Mr. Laudig, then chemist at the Buffalo Furnace Co., in a paper read before the Institute at the Chicago meeting, September, 1896.[2] This paper in my opinion furnished the first evidence that furnaces with wide tops, as they were then constructed, were not suited to working Mesabi ores. Mr. Laudig's experiments showed that Mesabi ores heated in the presence of reducing furnace-gases expanded in bulk rapidly, even before the charge had had time to be reduced in bulk by the consumption of the coke. This expansion caused the furnace to stick, from the wedging of the stock in the top of furnace, and explained the cause of top-slips, which were then very frequent when fine ores were used.

Furnacemen are not agreed as to the cause of the heavy slips and explosions formerly so frequent in using fine ores, but for-

[2] Action of Blast-Furnace Gases on Various Iron-Ores, *Trans.*, xxvi., 269.

tunately of less frequent occurrence in late years. I am inclined to the belief that, in cases of heavy explosions where the stock is thrown out of furnace, the trouble starts from a slip in the furnace, grinding coke to fine powder, which, coming into intimate contact with oxygen in the ore in a heated atmosphere, results in a dust-explosion. I do not approve of the closed furnace-tops, but think it advisable to provide liberal explosion-door area at the top of the furnace. The experience at the Shenango furnaces indicates that one or two large doors are more effective than several small ones having an equal total area. Nor do I think the theory held by the advocates of a closed-top, that holding back the force of the explosion by confining its escape to the usual avenues of gas-travel, lessens the bad effects, although it distributes it over a longer time. It is of no advantage to transfer the dust and material carried upward and usually thrown out the doors, to the gas-mains, dust-catchers and stoves. It certainly does not prevent slips and explosions, as may be testified by those in charge of plants where the closed-top is in use.

Our experience in handling furnaces with mixtures containing large portions of Mesabi ores has indicated the following points in practice. 1. Care in the selection of the ore in order to get the coarsest ore obtainable. 2. Adjusting the furnace-top and the lines to suit the ore-mixture. 3. Care in filling, placing the stock on the bell and lowering the stock into the furnace. 4. Arranging ample escape for explosions and bolting the hopper at least 15 ft. down the brick-work at the top of the furnace. 5. Avoid having the blast off as much as possible, for the reason that long stops are particularly liable to cause slips. 6. Necessity for increasing the amount of slag produced, our best results having been obtained with from 1,300 to 1,350 lb. per ton of pig iron, and using dolomite as a part of the flux which gives a good fluid slag and keeps hearth and lower bosh-walls clean, even though the analysis shows a slag quite basic. 7. Keeping the furnace working hot and gray, because the quantity of flue-dust produced invariably increases if the furnace works cold and the gases become thin. 8. Sufficient engine-, stove- and boiler-capacity to enable the furnacemen to "run" the furnace and not have the furnace "run" them.

Note on the Further Discussion of the Physics of Cast-Iron.

BY WILLIAM R. WEBSTER, PHILADELPHIA, PA.

(Atlantic City Meeting, February, 1904.)

At the Florida meeting of the Institute, in March, 1895, I presented a Note on a Proposed Scheme for the Study of the Physics of Cast-Iron,[1] giving an outline which, for convenience of reference, I here repeat.

Suggested Lines for Discussion and Investigation.

I.—Correspondence between chemical composition and melting-point, fluidity, shrinkage, fracture, chill, micro-structure, and other physical properties.

II.—Influence of:

1. Cupola-mixture, use of steel and other scrap, oxidized or clean material,	on	A. Fracture.
2. Manner of melting, flux, etc.,		B. Micro-structure.
3. Casting-temperature,		C. Physical properties.
4. Manner of handling melted metal and method of casting,		D. Shrinkage.
5. Size and form of casting,		E. Chill.
6. Kind of mold, green sand (under different conditions of ramming, amount of moisture, and skin-dried), dry sand, loam, and chills,		F. Residual stress.
7. Rate and mode of cooling castings,		G. Condition and quantity of carbon and other elements.
8. Manner and temperature of heating for annealing,		
9. Additions of nickel or aluminum,		

III.—Segregation as affected by: 1. Composition; 2. Casting-temperature; 3. Rate of cooling; 4. Size and shape of casting.

IV.—Blow-holes, their volume and position as affected by: 1. Composition; 2. Casting-temperature; 3. Casting-pressure; 4. Rate of cooling; 5. Size and shape of casting; 6. Special additions.

It was hoped that a discussion might thus be elicited, similar in interest and value to the discussion of the physics of steel, which had been so widely and gratefully welcomed by students and practitioners in that department. That this hope was not

[1] *Trans.*, xxv., 84.

disappointed is abundantly shown by the contributions to our *Transactions*, under the head of "Physics of Cast-Iron," of Messrs. F. E. Thompson, Pottstown, Pa. (xxv., 964); S. M. Vauclain, Baldwin Locomotive Works, Philadelphia, Pa. (xxv., 967); H. V. Wille, Philadelphia, Pa. (xxv., 969); C. R. Baird & Co., Philadelphia, Pa. (xxv., 971, and xxvi., 997); Thomas D. West, Sharpsville, Pa. (xxv., 972); William C. Henderson, Philadelphia, Pa. (xxv., 974); F. Schumann, Tacony Iron & Metal Co., Philadelphia, Pa. (xxv., 975); E. D. Estrada, Philadelphia, Pa. (xxv., 979); Asa W. Whitney, Philadelphia, Pa. (xxv., 980, and xxvi., 1004); Henry D. Hibbard, Highbridge, N. J. (xxv., 988); Prof. R. C. Carpenter, Ithaca, N. Y. (xxvi., 1002); William Kent, Passaic, N. J. (xxvi., 1014); Leonard Waldo, Bridgeport, Conn. (xxvi., 1017); George Morison, Chicago, Ill. (xxvi., 1019); A. E. Outerbridge, Jr., Philadelphia, Pa. (xxvi., 1019, 1023); David Townsend, Philadelphia, Pa. (xxvi., 1021).

To these should be added the separate papers, contained in vol. xxvi., of Messrs. Thomas D. West, A. J. Rossi, and Alexander E. Outerbridge, Jr., and the paper of Dr. Persifor Frazer (xxviii., 613) on The Kytchtym Medal, and the notable discussion thereof (xxviii., 848).

For five years past, however, our *Transactions* have contained little on this subject; and having been requested by the Council to aid in an attempt to revive the interesting discussion, I have thought that I could not do better than repeat the course which before proved so successful. I have therefore urged those who took part at that time to contribute their present views, recent investigations, and any other statements which they may deem appropriate, in order to bring the matter up to date. I trust that others also will give their assistance, and especially that those who have been working on the preparation of various commercial specifications for cast-iron, methods of testing, etc., will present such specifications for discussion, and give the reasons and methods which have led to their adoption.

A point of peculiar modern importance, is the classification of pig-iron by its chemical analysis, which involves, of course, the relation between its chemical constitution and its physical properties. Since our former discussion, the use of machine-cast sandless pig-iron has forced the foundrymen to give up, to

a large extent, the grading of the pig by fracture and to depend upon the chemical analysis alone. This, in turn, has brought them to rely on the relation between the chemical constitution of the mixture and the physical properties of the finished casting. They are also now more inclined to discuss the information they have acquired on these subjects, and to feel that they will be well repaid by the assistance received from such an interchange of ideas.

Notes on the Physics of Cast-Iron.

BY RICHARD MOLDENKE, NEW YORK CITY.

(Atlantic City Meeting, February, 1904.)

In crowding the recent mass of work on the physics of cast-iron into the compass of a short review, I cannot do better than to follow the lines of Mr. W. R. Webster's suggestion, made at the Florida meeting of the Institute, March, 1895.[1] The classification of the properties of the metal, and the influences to which it is subjected during the various processes it undergoes, is quite complete; and as Mr. Webster gives it again in full in another paper at the present meeting, a simple reference to it now will be sufficient.

I. *Correspondence Between Chemical Composition, and Melting-Point, Fluidity, Shrinkage, Fracture, Chill, Micro-Structure, and Other Physical Properties.*

Melting-Point.—The melting-point of cast-iron had long been a subject of controversy among foundrymen. They knew that a hard iron melted faster than a soft one, but this effect was occasionally disputed, and the reasons given were unsatisfactory, but could not be well controverted. It was my good fortune to take the actual melting-temperatures of some 70 pig-irons and castings; the Le Chatelier pyrometer, suitably protected, being introduced into the interior, and the iron melted off in a specially-constructed cupola. The irons had been previously analyzed by taking the borings at the point of the intro-

[1] *Trans.*, xxv., 984.

duction of the thermo-couple. The results show a regularly-ascending scale of melting-temperatures, as the amount of combined carbon decreases. That is, perfect white iron melted at the lowest temperature, and gray iron, containing hardly a sign of carbon in combination, resisted the melting-heat the longest. This means that a 4-per-cent. steel (if we may thus denominate a white iron which has no graphite in admixture) melted faster than a 0.10-carbon steel, containing nearly 4 per cent. of graphite mechanically intermixed, or gray iron. This conclusion seems natural in the light of our knowledge of steel, and, indeed, it seems to have been generally accepted as conclusive.

Fluidity.—The fluidity of a cast-iron is known to depend, in some measure, upon the phosphorus-content, yet little has been done in this field, the observations having been accepted apparently without question. It were well if a series of experiments could be made to test this assumption, which might throw additional light on the subject, of value in other directions. Oxidation has an important bearing on the fluidity of the metal, and a bath of highly-oxidized metal loses its fluidity so rapidly by the lowering of the temperature resulting from tapping and carrying, that the ladles "skull" and the molds are short-poured; an effect entirely independent of phosphorus-content of the metal, be it high or low.

Shrinkage and Contraction.—On the question of contraction, and its relation to the chemical composition, no general law can be laid down. A dead-white iron is supposed to contract a little more than 0.25 in. to the foot; a dead-gray iron, a little less than 0.125 in. to the foot. Between these two typical extremes in cast-iron, there is a large series of commercial products in which the amount of contraction shades from one to the other. Unfortunately for methods of measuring contraction and judging the composition therefrom, the casting-temperature plays sad havoc with the results. A hot iron always has a greater contraction than if the metal were held in the ladle for some time before pouring. Just why this is the case has not been definitely settled. The distinction between contraction and shrinkage should be noted. These terms are invariably confounded by the gray-iron founder. Shrinkage takes place at the moment of set, the metal becoming spongy.

Contraction takes place after the moment of set, the red-hot casting slowly contracting as it cools.

Fracture.—The fracture of an iron is directly related to its chemical composition; yet it is no criterion of what the iron will be, after remelting and casting under normal conditions. The furnaceman, who gets more money for the large-grained, black, open-fractured iron than for the close and light-gray variety, naturally wants to make only the former, hence all kinds of methods are adopted to achieve this result. Look over the metal-yard of a foundry and note the heavy sections that pig-iron has come to assume; also the signs of the sand which has been thrown on the molten metal in order to retard the cooling of the cast. The machine-cast pig-iron, which is now beginning to find its way, more and more, into the foundry, has helped to remove the last vestige of prejudice of the otherwise enlightened founder in the fracture-test. But long before this machine-cast metal was sold to the founder in large quantities, he had come to realize that the chemical specification alone was the best safeguard. To-day the founder, who buys by grade and demands a given fracture, gets his material, having the desired appearance, but with it, in addition, a nice assortment of compositions which would soon bring him to grief were it not for the custom of keeping a lot of irons always at hand and in the mixture. Meanwhile the founder who knows gets the pick of the market.

Chill.—The hotter the iron is cast, the deeper will be the chill. Unquestionably, the proportion of combined carbon left in a casting is at the bottom of this phenomenon, but why the deeper chill should go with the hotter iron, when exactly the opposite is expected, is a question still unanswered.

II. 1. *Cupola-Mixture.*—For the cupola-mixture, and, for that matter, the furnace-mixture also, the addition of steel-scrap has done much to improve the quality of the castings made in this way. It remains cast-iron, just the same, but the total carbon has been lowered considerably, which, with a practically-constant combined-carbon content, means the reduction of the graphite separated out, or, in other words, fewer planes of weakness in the metal. The fracture becomes lighter, as a consequence, and the contraction greater. If good results are

wanted, about 25 per cent. of steel-scrap is the maximum proportion allowable in the cupola. Above this quantity, great irregularity in the product is observed. For furnace-work, up to 30 per cent. can be used; but even here, the effect of oxidation becomes quite marked, shrinkage in the interior of the casting too pronounced, residual strains serious, and the molding-losses too much to permit. Nevertheless, the moderate use of steel is a good thing in the foundry, and should be encouraged.

When oxidized material is charged (that is to say, burnt iron grate-bars, salamanders, and the like, and, to a lesser extent, rusty material), the product will always be bad and were better left alone and put into sash-weights. Very rusty scrap is not detrimental if some silico-spiegel or ferromanganese be charged with the heat. In the absence of these alloys, however, plenty of limestone and a little fluor-spar to promote fluidity is beneficial.

II. 2. *Manner of Melting in the Foundry.*—Although the modern high-pressure production tends to injure the metal, blast-pressures are climbing, and coke is sacrificed to save time, not so much in order to crowd the pouring to the end of the day for convenience, but because labor-exactions have made it necessary to take away from the men as much as possible any excuse for loafing. Very little molding is done after the iron begins to run. As our pig-irons are not any too good to start with, and the melting-processes of the foundry, if anything, injures them still further, a change to conditions which would allow the iron to melt slowly, and be tapped out periodically during the day, would be welcomed. In those cases where the casting is continuous, greater pains can be taken with the charging, fluxing, etc., and possibly in the future, the iron may be tapped into a mixing-ladle as fast as melted, and thence taken to the molds. In this way, the iron would not remain in contact with the fuel very long and would have better properties. The European method is to be commended in this respect.

The effect of melting upon the physical structure of the iron, is fairly well marked in its subsequent manipulation in the machine-shop. Good hot iron in the cupola will not take up as much sulphur as will a cold heat. Hence, a difference in the structure is quickly noticed in planing-up a casting. In general

foundry-work, it is good practice to melt very hot, and to hold the iron until cool enough to cast safely. For special work this will not always hold true, as will be shown later.

With a large heat, poor slagging-out means dirty iron. At first it does not matter, but after 10 tons or more have been melted, the slag begins to be troublesome. It is better, therefore, to provide a slag-spout and to use it. The result will quickly be noted by a reduced scrap-pile in the trimming-room. Nothing is paid back so disagreeably as carelessness on the charging-platform.

The melting-process naturally has an effect on the composition. Manganese and silicon are burned out to some extent, sulphur is unfortunately increased, and the total carbon in the charge will be increased or diminished, according to circumstances. There is a tendency to hold a saturation-point fixed by the percentage of silicon present. If the total carbon is below this, it will be increased; if above, it may be diminished. In general, the less the silicon, the more the tendency of the metal to take up carbon.

II. 3. *Casting-Temperature.*—The casting-temperature is a very important item, for the reason that, while one class of work requires very cold iron, another must have it "piping-hot." It becomes really a question of surface-excellence for the casting to be poured. In one case, good chilling-effects must be obtained, together with absence of the slightest pin-holes, as for instance in the chilled roll, where the metal should be cast as hot as the chills can bear it without fusing. In another case, it may be necessary to have some machining done; in this case, the metal is poured cold, in order not to burn the sand of the mold to the casting. To return to the chilled roll, if this be cast with metal too cold, the casting sets too quickly, and the enormous casting-strains produced in the rim from its contraction upon a solidified interior always means trouble. Take the coefficient of expansion of white iron and note just how many inches this makes on a roll 36 in. in diameter, cooled from about 2,000° F. to ordinary temperatures. The inches thus lost in the circumference must be taken up by a suitable stretch in the metal there, and, in the case of a cold pour, a rupture along the whole face of the casting will result. If, on the other hand,

the roll be cast with very hot metal, the rim is at once chilled and set, while the interior is still molten; and by the time the interior sets, the solid rim has shrunk to some extent, and at least is sufficiently strong to withstand the internal resistance to final contraction without tearing apart.

II. 5. *Size and Form of Casting.*—The thickness of the iron in a casting, bears a direct relation to the rate of cooling, and, consequently, on its physical structure, as already explained. Therefore, in making mixtures, it is necessary to keep in mind the thin parts as well as the thick. Thus, a heavy dynamo-frame, for which a mixture containing 1.75 per cent. of silicon would be just right, may have a small thin bracket which requires machining. The silicon therefore must be raised to 2.15 per cent. in order that this part shall possess the proper composition, otherwise the machine-shop work may cost more than the additional price paid for the softer iron.

II. 6. *Kind of Mold, etc.*—The mold itself also has an effect, loam- and dry-sand methods allowing hotter iron to be used with good surface-results in those cases in which green sand would give a badly scabbed or skin-hardened casting. With a very heavy body, the effect is not so marked, for the reason that there is a subsequent annealing, which proceeds from within, outward.

II. 8. *Manner and Temperature of Annealing.*—The annealing of castings is a study by itself, of which we have still much to learn. The process consists in heating the material to be annealed to a high temperature, and then slowly cooling it. Where the sulphur-content is fairly low, and the edges of small castings are too hard for rapid machining, annealing them will be beneficial. With higher sulphur, it often becomes a question solely of scrapping the lot. Between the annealing-process applied to malleable castings, on the one hand, and gray iron, on the other, undoubtedly the special form of free carbon, known as temper-carbon, will be found to play an important part also in the gray-iron branch of the foundry industry.

II. 9. *Additions of Nickel or Aluminum.*—The use of aluminum in the foundry is becoming more limited than it was, for the reason that it has been found to be injurious to the harder irons; that is, for hard irons the remarkable property possessed

by the light metal of throwing out the graphite, militates against its usefulness. For the soft varieties of iron, a small quantity of aluminum, added when gases are feared, is very good, but, as above mentioned, proper precautions in melting should remove the necessity of this addition. On the other hand, the use of titanium-iron alloys is to be commended, because the titanium reacts directly with any oxygen or nitrogen present in solution, and, as a consequence, a purification takes place which cannot be overvalued. Increasing the strength of an iron 20 per cent., without remaining behind as an integral part of the metal, seems to be the function of this new candidate for foundry favors. With regard to the addition of other metals, such as nickel, cobalt, etc., but little is heard about them in the foundry. These metals, or their alloys, are too expensive, and, consequently, it is more profitable to use steel castings for material of a strength greater than that of cast-iron.

III. *Segregation.*—The subject of segregation in cast-iron is one which could well be studied more fully. Apart from the formation of graphite, which may be considered a sort of distributed segregation, we know very little of the peculiarities of iron phosphide, and still less of iron sulphide, so far as their tendency to segregate is concerned. On account of the lower melting-point of iron phosphide, globules of iron are occasionally found with a higher phosphorus-content than the rest of the metal, which have been pushed out of the main body, or have entered into the blow-holes in the interior of the casting. By far the most important subject for study, is the segregation of the sulphides, and in this field the microscope will eventually prove itself most valuable. Up to the present time, this indispensable adjunct to the laboratory has done little for cast-iron, partly because the methods of determining constituents other than carbon are practically wanting, and also, because our steel experts know from actual practice very little of cast-iron.

IV. *Blow-Holes.*—Blow-holes should not exist with good molding, good iron and good melting, and, generally speaking, they result from errors of manipulation. Where the sand has been rammed too hard, or the venting is imperfect, air is often imprisoned and sent about in bubbles, usually covered over by a skin of iron. If a rather large blow-hole exists near the sur-

face, the iron can often be "burned on," and the casting saved. The blow-holes to be feared are the small gas-pockets, such as those appearing on the surface of a chilled roll, practically pin-head in size, but, nevertheless, sufficient to condemn the work. In this case, the metal is oxidized to an extent that gas is liberated at the moment of set, and becomes imprisoned very near the surface, often causing disastrous results. The trouble with blow-holes is usually confined to the low-silicon range of cast-irons, and were it not for the fact that the addition of aluminum promotes the formation of graphite, this metal would form a valuable corrective. We must look to the future for relief, possible with ferro-titanium alloys, or the ferro-compounds of such metals as magnesium. If a large furnace heat gives indications of being heavily charged with oxides, the only safe way to act is to add plenty of ferro-silicon, rabble thoroughly, and cast all of the metal into pigs, which can be subsequently fed into the regular mixture in small amounts, until the supply of this unwelcome metal is used up.

The future of our studies along the lines indicated above, seems to lie in the devising of methods to control the chemical composition of cast-iron in cupola and furnace, irrespective of the nature of the pig and scrap charged. Thus, we should be able to eliminate sulphur, and perhaps phosphorus, at will. We should also be able to remove the last traces of oxidation, which has been due either to the blast-furnace or to the cupola-furnace at some previous smelting. With those desiderata, the field for the maker of foundry pig-iron will be greatly broadened, and many an iron-ore, now unsalable, will find its way into the market, to the lasting benefit of world's mineral resources.

The Standardization of Specifications for Iron and Steel: Recent Progress in America and England.

BY WILLIAM R. WEBSTER AND EDGAR MARBURG, PHILADELPHIA, PA.

(Atlantic City Meeting, February, 1904.)

THE desirability of bringing about greater uniformity in specifications governing iron and steel is generally recognized, and has found expression within recent years, in the efforts of numerous technical societies in that direction.

When the International Association for Testing Materials was organized at Zurich in 1895, a committee was appointed, charged as follows:

"On the basis of existing specifications, to seek methods and means for the introduction of international specifications for testing and inspecting iron and steel of all kinds."

The American representation on this international committee consisted originally of five, and now of eight members. In view of the magnitude and importance of the subject, the Executive Committee of the American Section of the International Association for Testing Materials, since incorporated as the American Society for Testing Materials, appointed a committee of 34 members, including the American members of the above-named international committee, to frame standard American specifications for iron and steel. This committee reported on specifications for (1) Structural Steel for Bridges and Ships; (2) Structural Steel for Buildings; (3) Open-Hearth Boiler-Plate and Rivet-Steel; (4) Steel Rails; (5) Steel Splice-Bars; (6) Steel Axles; (7) Steel Tires; (8) Steel Forgings; (9) Steel Castings; (10) Wrought-Iron.

These specifications were designed to be fairly representative of the best current American practice, and were adopted by letter-ballot of the Society in August, 1901.

The leading engineering societies have participated at various times in the discussion of these specifications and have lent valuable assistance through the appointment of special committees on like, or closely related, subjects.

The American Society of Civil Engineers discussed the spec-

ifications for Structural Steel for Bridges and Ships, and appointed a Committee on Steel Rails, to report on Sections, Methods of Manufacture, Specifications, and Methods of Testing and Inspection.

The American Society of Mechanical Engineers discussed the specifications for Boiler-Plate and Rivet-Steel, Steel Forgings and Steel Castings, and a committee of that society presented last year, a report on these specifications, subject to revision, recommending certain modifications.

The American Master Mechanics' Association appointed a committee to report on the specifications for Steel Axles and Steel Forgings. This committee made a preliminary report last year, and has the matter still under consideration.

The American Railway Engineering and Maintenance of Way Association appointed separate committees to report on specifications for Bridge Materials and Steel Rails. These specifications, which differ somewhat from those of the American Society for Testing Materials, were adopted by vote of the Association, the former in 1903 and the latter in 1902. The Committee on Steel Rails has been instructed, however, to consider certain proposed modifications.

The American Institute of Mining Engineers has co-operated in this work by discussing the specifications for Steel Rails, Steel Forgings and Steel Castings, and is now asked to assist again, by including in the discussion on the "Physics of Cast-Iron" the specifications proposed by the Committee on "Standard Specifications for Cast-Iron and Finished Castings" of the American Society for Testing Materials. These specifications, which will be presented at this meeting by the Chairmen of the Sub-Committees concerned, comprise,—(1) Specifications and Grading of Pig-Iron; (2) Cast-Iron Water- and Gas-Pipe; (3) Cylinder-Castings; (4) Car-Wheels; (5) Malleable Iron; (6) General Castings and Methods of Testing.

The above presentation, covers in brief outline, the recent history of the movement on the part of technical societies in this country in the direction of the standardization of specifications for iron and steel. The desirability of arriving at some uniform basis of reasonableness, in specifications governing iron and steel products, is too obvious for extended argument. The ordering of material which is to be subjected to similar or iden-

tical conditions of service under a great multiplicity of specifications, differing often more in the letter than in the spirit, is harassing to the manufacturer and of no real advantage to the consumer. The latter should recognize that any unnecessary burden imposed on the manufacturer must necessarily react in the end upon the consumer, and increase the average cost of the product without attendant gain in average quality. On the other hand, the manufacturer should recognize his obligation to spare no reasonable effort or expense to ensure a high-grade product of uniform quality, and should evince a willingness to provide adequate facilities for testing.

The existing differences between the leading specifications framed within recent years are, in the main, on matters of minor importance, and what has been done has resulted in a considerable clearing of the atmosphere. That further efforts will be put forth to reconcile the remaining differences as far as possible, cannot be doubted. If the task be approached in a fair and open spirit of compromise between interests whose divergence, broadly viewed, is more apparent than real, all parties will be the gainers. If this work be promptly initiated, it is not too much to hope that American standard specifications, approved by the leading technical societies, and covering the principal iron and steel products, will be available for presentation at the Seventh Session of the International Railway Congress to be convened in Washington, D.C., in May, 1905, and that they will prove an important step towards the ultimate realization of international standard specifications.

Turning now to a brief review of the present situation in England, it will be seen that much the same agencies have been set at work there, and that the outlook is in some respects even more auspicious than in America.

The movement in England had its origin in the appointment by the Institution of Civil Engineers of a Committee on Engineering Standards, which began its labors in 1901. The Institution of Civil Engineers secured the co-operation of the Institution of Mechanical Engineers, the Institution of Naval Architects, the Iron and Steel Institute and the Institution of Electrical Engineers, and the appointment by these several societies of representatives on the committee. According to a recently published announcement these "five leading technical

institutions are thus actively and financially supporting and controlling the operations of this important movement."

In recognition of the great national importance of this undertaking, the British Government has lately authorized a grant of £3,000 towards the necessary expenses, and the India Government has agreed to contribute an additional sum of £1,000.

The War Office and Admiralty Departments, who are among the largest users of iron and steel, are actively co-operating in this work, and there are now more than 24 representatives of the Government on the various Sub-Committees. Representation on these Committees has also been accorded to numerous engineering, scientific and trade organizations, to the International Association for Testing Materials, as well as to the leading manufacturing- and consuming-interests.

The scope of the field, which this Committee on Engineering Standards proposes to cover, may be judged from the following list of Committees and Sub-Committees:

1. Sections used in Ship Building (11 members).
    (a) Sub-Committee on Tests for Iron and Steel Material Used in the Construction of Ships and their Machinery (23 members).
2. Bridges and General Building Construction (12 members).
3. Railway Rolling-Stock Underframes (13 members).
4. Locomotives (28 members).
    (a) Sub-Committee on Component Parts and Tires (14 members);
    (b) Sub-Committee on Steel Plates (7 members);
    (c) Sub-Committee on Tires, Axles and Springs (6 members);
    (d) Sub-Committee on Copper and its Alloys (6 members).
5. Rails (22 members).
    (a) Section on Railway Rails (11 members);
    (b) Section on Tramway Rails (4 members).
6. Electrical Plant (22 members).
    (a) Sub-Committee on Generators, Transformers and Motors (13 members);
    (b) Sub-Committee on Temperatures of Insulation Materials (5 members);

(c) Sub-Committee on Cables and Conduits (11 members);

(d) Sub-Committee on Telegraphs and Telephones (7 members).

7. Screw-Threads and Limit-Gauges (26 members).

(a) Sub-Committee on Screw-Threads (26 members);

(b) Sub-Committee on Limit-Gauges (17 members).

8. Pipe-Flanges (12 members).

9. Cement (17 members).

Three of the above-named committees have considered independently, the question of "Standard Sections for Rolled Iron and Steel," and have held joint-meetings and have agreed upon standard sections which have been adopted and published.

It will be seen from the foregoing, that the English Engineering Standards Committee is operating on lines quite similar to those adopted by the American Society for Testing Materials. Both have recognized the desirability of giving fair representations on their committees to all parties in interest, including the manufacturers. The work is divided among numerous sub-committees whose reports are submitted to the parent committee, which, in turn, makes its report to the technical societies which it represents.

The work in England is much facilitated, however, first, from the fact that the five leading technical societies are co-operating through the medium of a single, jointly-appointed committee; and second, by reason of the liberal financial support which the Government has lent to the undertaking.

Notwithstanding certain disadvantages, however, the progress towards the standardization of specifications, that has been made in this country during the past few years, is distinctly creditable; and it is confidently expected that the moral and financial support for continuing this work, along broader lines, will be duly forthcoming.

Standard Specifications for Pig-Iron and Iron Products.

(Printed in Connection with the Discussion of this Subject at the Atlantic City Meeting, February, 1904.)

THE following specifications, proposed by a sub-committee of the American Society for Testing Materials, are still to be adopted and subsequently submitted to the International Association for Testing Materials.

Standard Specifications for Pig-Iron and Castings.

I. ANALYSIS.—It is recommended that all purchases be made by analysis.

II. SAMPLING.—In contracts where pig-iron is sold by chemical analysis, each carload, or its equivalent, shall be considered as a unit. At least one pig shall be selected at random from each two tons of every carload, and so as to fairly represent it.

Drillings shall be taken so as to fairly represent the fracture surface of each pig, and the sample analyzed shall consist of an equal quantity of drillings from each pig, well mixed and ground before analysis.

III. ALLOWANCES AND PENALTIES.—In all contracts, in the absence of a definite understanding to the contrary, a variation of 10 per cent. of silicon, either way, and of .01 in sulphur above the standard is allowed. A deficiency of over 10 per cent. in the silicon, up to 20 per cent., and a further increase in sulphur up to .01 over the above allowance subjects the shipment to a penalty of 1 per cent. in the price for each element so affected.

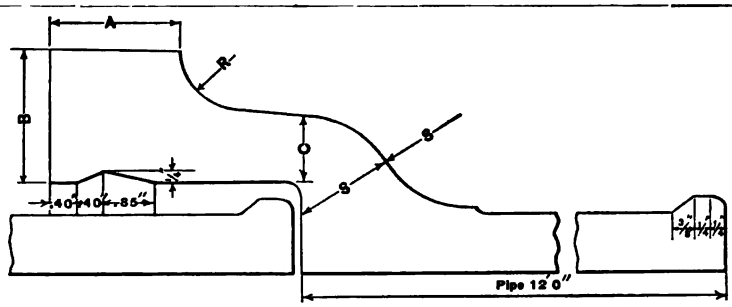
IV. BASE ANALYSIS OF GRADES.—In the absence of specifications the following numbers, known to the trade, shall represent the appended analyses for standard grades of foundry pig-iron, irrespective of the fracture, and subject to allowances and penalties as above.

Grade.	Silicon Per Cent.	Sulphur Per Cent.
No. 1,	2.75	0.035
No. 2,	2.25	0.045
No. 3,	1.75	0.055
No. 4,	1.25	0.065

Cast-Iron Pipe and Special Castings.

I. DESCRIPTION OF PIPES.—The pipes shall be made with hub- and spigot-joints, and shall accurately conform to the dimensions given in Tables Nos. I. and II. They shall be straight and shall be true circles in section, with their inner and outer surfaces concentric, and shall be of the specified dimensions in outside diameter. They shall be at least 12 feet in length, exclusive of socket. For pipes of each size from 4-inch to 24-inch, inclusive, there shall be two standards of outside diameter, and for pipes from 30-inch to 60-inch, inclusive, there shall be four standards of outside diameter, as shown by Table No. II.

TABLE I.—*General Dimensions of Cast-Iron Pipes.*



Nominal Diam. Inches.	Classes.	Actual Outside Diam. Inches.	Diam. of Sockets.		Depth of Sockets.		A.	B.	C.
			Pipe. Inches.	Special Castings. Inches.	Pipe. Inches.	Special Castings. Inches.			
4	A—B	4.80	5.60	5.70	3.5	4.0	1.5	1.3	.65
4	C—D	5.00	5.80	5.70	3.5	4.0	1.5	1.3	.65
6	A—B	6.90	7.70	7.80	3.5	4.0	1.5	1.4	.70
6	C—D	7.10	7.90	7.80	3.5	4.0	1.5	1.4	.70
8	A—B	9.05	9.85	10.00	4.0	4.0	1.5	1.5	.75
8	C—D	9.30	10.10	10.00	4.0	4.0	1.5	1.5	.75
10	A—B	11.10	11.90	12.10	4.0	4.0	1.5	1.5	.75
10	C—D	11.40	12.20	12.10	4.0	4.0	1.5	1.6	.80
12	A—B	13.20	14.00	14.20	4.0	4.0	1.5	1.6	.80
12	C—D	13.50	14.30	14.20	4.0	4.0	1.5	1.7	.85
14	A—B	15.30	16.10	16.10	4.0	4.0	1.5	1.7	.85
14	C—D	15.65	16.45	16.45	4.0	4.0	1.5	1.8	.90
16	A—B	17.40	18.40	18.40	4.0	4.0	1.75	1.8	.90
16	C—D	17.80	18.80	18.80	4.0	4.0	1.75	1.9	1.00
18	A—B	19.50	20.50	20.50	4.0	4.0	1.75	1.9	.95
18	C—D	19.92	20.92	20.92	4.0	4.0	1.75	2.1	1.05
20	A—B	21.60	22.60	22.60	4.0	4.0	1.75	2.0	1.00
20	C—D	22.06	23.06	23.06	4.0	4.0	1.75	2.3	1.15
24	A—B	25.80	26.80	26.80	4.0	4.0	2.00	2.1	1.05
24	C—D	26.32	27.32	27.32	4.0	4.0	2.00	2.5	1.25
30	A	31.74	32.74	32.74	4.5	4.5	2.00	2.3	1.15
30	B	32.00	33.00	33.00	4.5	4.5	2.00	2.3	1.15
30	C	32.40	33.40	33.40	4.5	4.5	2.00	2.6	1.32
30	D	32.74	33.74	33.74	4.5	4.5	2.00	3.0	1.50
36	A	37.96	38.96	38.96	4.5	4.5	2.00	2.5	1.25
36	B	38.30	39.30	39.30	4.5	4.5	2.00	2.8	1.40
36	C	38.70	39.70	39.70	4.5	4.5	2.00	3.1	1.60
36	D	39.16	40.16	40.16	4.5	4.5	2.00	3.4	1.80
42	A	44.20	45.20	45.20	5.0	5.0	2.00	2.8	1.40
42	B	44.50	45.50	45.50	5.0	5.0	2.00	3 0	1.50
42	C	45.10	46.10	46.10	5.0	5.0	2.00	3.4	1.75
42	D	45.58	46.58	46.58	5.0	5.0	2.00	3.8	1.95
48	A	50.50	51.50	51.50	5.0	5.0	2.00	3.0	1.50
48	B	50.80	51.80	51.80	5.0	5.0	2.00	3.3	1.65
48	C	51.40	52.40	52.40	5.0	5.0	2.00	3.8	1.95
48	D	51.98	52.98	52.98	5.0	5.0	2.00	4.2	2.20
54	A	56.66	57.66	57.66	5.5	5.5	2.25	3.2	1.60
54	B	57.10	58.10	58.10	5.5	5.5	2.25	3.6	1.80
54	C	57.80	58.80	58.80	5.5	5.5	2.25	4.0	2.15
54	D	58.40	59.40	59.40	5.5	5.5	2.25	4.4	2.45
60	A	62.80	63.80	63.80	5.5	5.5	2.25	3.4	1.70
60	B	63.40	64.40	64.40	5.5	5.5	2.25	3.7	1.90
60	C	64.20	65.20	65.20	5.5	5.5	2.25	4.2	2.25
60	D	64.82	65.82	65.82	5.5	5.5	2.25	4.7	2.60

TABLE II.—*Standard Thicknesses and Weights of Cast-Iron Pipe 12 Feet in Length, Exclusive of Socket.*

Inches.	Class A. 100 Ft. Head. 43 Lb. Pressure.			Class B. 200 Ft. Head. 86 Lb. Pressure.			Class C. 300 Ft. Head. 130 Lb. Pressure.			Class D. 400 Ft. Head. 173 Lb. Pressure.			Inches.
Inside Diam.	Thickness. Inches.	Weight Per		Thickness. Inches.	Weight Per		Thickness. Inches.	Weight Per		Thickness. Inches.	Weight Per		Inside Diam.
		Foot.	Length.		Foot.	Length.		Foot.	Length.		Foot.	Length.	
4	.40	20.0	240	.45	21.7	260	.48	23.3	280	.52	25.0	300	4
6	.44	30.8	370	.48	33.3	400	.51	35.8	430	.55	38.3	460	6
8	.46	42.9	515	.51	47.5	570	.56	52.1	625	.60	55.8	670	8
10	.50	57.1	685	.57	63.8	765	.62	70.8	850	.68	76.7	920	10
12	.54	72.5	870	.62	82.1	985	.68	91.7	1,100	.75	100.0	1,200	12
14	.57	89.6	1,075	.66	102.5	1,230	.74	116.7	1,400	.82	129.2	1,550	14
16	.60	108.3	1,300	.70	125.0	1,500	.80	143.8	1,725	.89	158.3	1,900	16
18	.64	129.2	1,550	.75	150.0	1,800	.87	175.0	2,100	.96	191.7	2,300	18
20	.67	150.0	1,800	.80	175.0	2,100	.92	208.3	2,500	1.03	229.2	2,750	20
24	.76	204.2	2,450	.89	233.3	2,800	1.04	279.2	3,350	1.16	306.7	3,680	24
30	.88	291.7	3,500	1.03	333.3	4,000	1.20	400.0	4,800	1.37	450.0	5,400	30
36	.99	391.7	4,700	1.15	454.2	5,450	1.36	545.8	6,550	1.58	625.0	7,500	36
42	1.10	512.5	6,150	1.28	591.7	7,100	1.54	716.7	8,600	1.78	825.0	9,900	42
48	1.26	666.7	8,000	1.42	750.0	9,000	1.71	908.3	10,900	1.96	1,050.0	12,600	48
54	1.35	800.0	9,600	1.55	933.3	11,200	1.90	1,141.7	13,700	2.23	1,341.7	16,100	54
60	1.39	916.7	11,000	1.67	1,104.2	13,250	2.00	1,341.7	16,100	2.38	1,583.3	19,000	60

The above weights are for 12 feet laying lengths and standard sockets; proportionate allowance to be made for any variation therefrom.

All pipes having the same outside diameter shall have the same inside diameter at both ends. The inside diameter of the lighter pipes of each standard outside diameter shall be gradually increased for a distance of about 6 inches from each end of the pipe, so as to obtain the required standard thickness and weight for each size and class of pipe.

Pipes whose standard thickness and weight are intermediate between the classes in Table No. II. shall be made of the same outside diameter as the next heavier class. Pipes whose standard thickness and weight are less than shown by Table No. II. shall be made of the same outside diameter as the Class A pipes, and pipes whose thickness and weight are more than shown by Table No. II. shall be made of the same outside diameter as the Class D pipes.

For pipes 4-inch to 12-inch, inclusive, one class of special castings shall be furnished, made from Class D pattern. Those having spigot-ends shall have outside diameters of spigot-ends midway between the two standards of outside diameter as shown by Table No. II., and shall be tapered back for a distance of 6 inches. For pipes from 14-inch to 24-inch, inclusive, two classes of special castings shall be furnished, Class B special castings with Classes A and B pipes, and Class D special castings with Classes C and D pipes, the former to be stamped "AB" and the latter to be stamped "CD." For pipes 30-inch to 60-inch, inclusive, four classes of special castings shall be furnished, one for each class of pipe, and shall be stamped with the letter of the class to which they belong.

II. ALLOWABLE VARIATION IN DIAMETER OF PIPES AND SOCKETS.—Especial care shall be taken to have the sockets of the required size. The sockets and spigots will be tested by circular gauges, and no pipe will be received which is

defective in joint room from any cause. The diameters of the sockets and the outside diameters of the bead ends of the pipes shall not vary from the standard dimensions by more than .06 of an inch for pipes 16 inches or less in diameter; .08 of an inch for 18-inch, 20-inch and 24-inch pipes; .10 of an inch for 30-inch, 36-inch and 42-inch pipes; .12 of an inch for 48-inch, and .15 of an inch for 54-inch and 60-inch pipes.

III. ALLOWABLE VARIATION IN THICKNESS.—For pipes whose standard thickness is less than 1 inch, the thickness of metal in the body of the pipe shall not be more than .08 of an inch less than the standard thickness, and for pipes whose standard thickness is 1 inch or more, the variation shall not exceed .10 of an inch, except that for spaces not exceeding 8 inches in length in any direction, variations from the standard thickness of .02 of an inch in excess of the allowance above given shall be permitted.

For special castings of standard patterns a variation of 50 per cent. greater than allowed for straight pipe shall be permitted.

IV. DEFECTIVE SPIGOTS MAY BE CUT.—Defective spigot-ends on pipes 12 inches or more in diameter may be cut off in a lathe and a half-round, wrought-iron band shrunk into a groove cut in the end of the pipe. Not more than 12 per cent. of the total number of accepted pipes of each size shall be cut and banded, and no pipe shall be banded which is less than 11 feet in length, exclusive of the socket.

In case the length of a pipe differs from 12 feet, the standard weight of the pipe given in Table No. II. shall be modified in accordance therewith.

V. SPECIAL CASTINGS.—All special castings shall be made in accordance with the cuts and the dimensions given in the table forming a part of these specifications.

The diameters of the sockets and the external diameters of the bead-ends of the special castings shall not vary from the standard dimensions by more than .12 of an inch for castings 16 inches or less in diameter; .15 of an inch for 18-inch, 20-inch and 24-inch; .20 of an inch for 30-inch, 36-inch and 42-inch, and .24 of an inch for 48-inch, 54-inch and 60-inch. These variations apply only to special castings made from standard patterns.

The flanges on all manhole castings and manhole covers shall be faced true and smooth, and drilled to receive bolts of the sizes given in the tables. The manufacturer shall furnish and deliver all bolts for bolting on the manhole covers, the bolts to be of the sizes shown on plans and made of the best quality of mild steel, with hexagonal heads and nuts and sound, well-fitting threads.

VI. MARKING.—Every pipe and special casting shall have distinctly cast upon it the initials of the maker's name. When cast especially to order, each pipe and special casting larger than 4-inch may also have cast upon it figures showing the year in which it was cast and a number signifying the order in point of time in which it was cast, the figures denoting the year being above and the number below, thus:

1901	1901	1901
1	2	3

etc., also any initials, not exceeding four, which may be required by the purchaser. The letters and figures shall be cast on the outside and shall be not less than 2 inches in length and $\frac{1}{4}$ of an inch in relief for pipes 8 inches in diameter and larger. For smaller sizes of pipes the letters may be 1 inch in length. The weight and the class letter shall be conspicuously painted in white on the inside of each pipe and special casting after the coating has become hard.

VII. ALLOWABLE PERCENTAGE OF VARIATION IN WEIGHT.—No pipe shall be accepted the weight of which shall be less than the standard weight by more than 5 per cent. for pipes 16 inches or less in diameter, and 4 per cent. for pipes more than 16 inches in diameter, and no excess above the standard weight of more than the given percentages for the several sizes shall be paid for. The total weight to be paid for shall not exceed for each size and class of pipe received, the sum of the standard weights of the same number of pieces of the given size and class by more than 2 per cent.

No special casting shall be accepted the weight of which shall be less than the standard weight by more than 10 per cent. for pipes 12 inches or less in diameter, and 8 per cent. for larger sizes, except that curves, Y pieces and breeches pipe may be 12 per cent. below the standard weight, and no excess above the standard weight of more than the above percentages for the several sizes will be paid for. These variations apply only to castings made from the standard patterns.

VIII. QUALITY OF IRON.—All pipes and special castings shall be made of cast-iron of good quality, and of such character as shall make the metal of the castings strong, tough and of even grain, and soft enough to satisfactorily admit of drilling and cutting. The metal shall be made without any admixture of cinder-iron or other inferior metal, and shall be remelted in a cupola or air-furnace.

IX. TESTS OF MATERIAL.—Specimen bars of the metal used, each being 26 inches long by 2 inches wide and 1 inch thick, shall be made without charge as often as the engineer may direct, and, in default of definite instructions, the contractor shall make and test at least one bar from each heat or run of metal. The bars, when placed flatwise upon supports 24 inches apart and loaded in the center, shall, for pipes 12 inches or less in diameter, support a load of 1,900 pounds and show a deflection of not less than .30 of an inch before breaking, and for pipes of sizes larger than 12 inches shall support a load of 2,000 pounds and show a deflection of not less than .32 of an inch. The contractor shall have the right to make and break three bars from each heat or run of metal, and the test shall be based upon the average results of the three bars. Should the dimensions of the bars differ from those above given, a proper allowance therefor shall be made in the results of the tests.

X. CASTING OF PIPES.—The straight pipes shall be cast in dry sand molds in a vertical position. Pipes 16 inches or less in diameter shall be cast with the hub end up or down, as specified in the proposal. Pipes 18 inches or more in diameter shall be cast with the hub end down.

The pipes shall not be stripped or taken from the pit while showing color of heat, but shall be left in the flasks for a sufficient length of time to prevent unequal contraction by subsequent exposure.

XI. QUALITY OF CASTINGS.—The pipes and special castings shall be smooth, free from scales, lumps, blisters, sand holes and defects of every nature which unfit them for the use for which they are intended. No plugging or filling will be allowed.

XII. CLEANING AND INSPECTION.—All pipes and special castings shall be thoroughly cleaned and subjected to a careful hammer inspection. No casting shall be coated unless entirely clean and free from rust, and approved in these respects by the engineer immediately before being dipped.

XIII. COATING.—Every pipe and special casting shall be coated inside and out with coal-tar pitch varnish. The varnish shall be made from coal tar. To this material sufficient oil shall be added to make a smooth coating, tough and tenacious when cold, and not brittle, nor with any tendency to scale off.

Each casting shall be heated to a temperature of 300 degrees Fahrenheit immediately before it is dipped, and shall possess not less than this temperature at the time it is put in the vat. The ovens in which the pipes are heated shall be so arranged that all portions of the pipe shall be heated to an even temperature. Each casting shall remain in the bath at least five minutes.

The varnish shall be heated to a temperature of 300 degrees Fahrenheit (or less if the engineer shall so order), and shall be maintained at this temperature during the time the casting is immersed.

Fresh pitch and oil shall be added when necessary to keep the mixture at the proper consistency, and the vat shall be emptied of its contents and refilled with fresh pitch when deemed necessary by the engineer. After being coated the pipes shall be carefully drained of the surplus varnish. Any pipe or special casting that is to be recoated shall first be thoroughly scraped and cleaned.

XIV. HYDROSTATIC TEST.—When the coating has become hard, the straight pipes shall be subjected to a proof by hydrostatic pressure, and, if required by the engineer, they shall also be subjected to a hammer test under this pressure.

The pressure to which the different sizes and classes of pipes shall be subjected are as follows:—

	20-Inch Diameter and Larger.	Less than 20-Inch Diameter.
	Pounds per Sq. Inch.	Pounds per Sq. Inch.
Class A pipe.....	150	300
Class B pipe.....	200	300
Class C pipe.....	250	300
Class D pipe.....	300	300

XV. WEIGHING.—The pipes and special castings shall be weighed for payment under the supervision of the engineer after the application of the coal-tar pitch varnish. If desired by the engineer, the pipes and special castings shall be weighed after their delivery, and the weights so ascertained shall be used in the final settlement, provided such weighing is done by a legalized weighmaster. Bids shall be submitted and a final settlement made up on the basis of a ton of 2,000 pounds.

XVI. CONTRACTOR TO FURNISH MEN AND MATERIALS.—The contractor shall provide all tools, testing-machines, materials and men necessary for the required testing, inspection and weighing at the foundry of the pipes and special castings; and, should the purchaser have no inspector at the works, the contractor shall, if required by the engineer, furnish a sworn statement that all of the tests have been made as specified, this statement to contain the results of the tests upon the test-bars.

XVII. POWER OF ENGINEER TO INSPECT.—The engineer shall be at liberty at all times to inspect the material at the foundry, and the molding, casting and coating of the pipes and special castings. The forms, sizes, uniformity and conditions of all pipes and other castings herein referred to shall be subject to his inspection and approval, and he may reject, without proving, any pipes or other casting which is not in conformity with the specifications or drawings.

XVIII. INSPECTOR TO REPORT.—The inspector at the foundry shall report daily to the foundry office all pipes and special castings rejected, with the causes for rejection.

XIX. CASTINGS TO BE DELIVERED SOUND AND PERFECT.—All the pipes and other castings must be delivered in all respects sound and conformable to these

specifications. The inspection shall not relieve the contractor of any of his obligations in this respect, and any defective pipe or other castings which may have passed the engineer at the works or elsewhere shall be at all times liable to rejection when discovered, until the final completion and adjustment of the contract; provided, however, that the contractor shall not be held liable for pipes or special castings found to be cracked after they have been accepted at the agreed point of delivery. Care shall be taken in handling the pipes not to injure the coating, and no pipes or other material of any kind shall be placed in the pipes during transportation or at any time after they receive the coating.

XX. DEFINITION OF THE WORD "ENGINEER."—Wherever the word "engineer" is used herein it shall be understood to refer to the engineer or inspector acting for the purchaser and to his properly authorized agents, limited by the particular duties intrusted to them.

Locomotive Cylinders.

I. PROCESS OF MANUFACTURE.—Locomotive cylinders shall be made from good quality of close-grained gray iron cast in a dry sand mold.

II. CHEMICAL PROPERTIES.—Drillings taken from test-pieces cast as hereafter mentioned shall conform to the following limits in chemical composition: ·

Silicon, ...	from 1.25 to 1.75 per cent.
Phosphorus, ...	not over .9 per cent.
Sulphur, ...	not over .10 per cent.

III. PHYSICAL PROPERTIES.—The minimum physical qualities for cylinder iron shall be as follows:

The "Arbitration Test-Bar," 1¼ inches in diameter, with supports 12 inches apart, shall have a transverse strength not less than 2,700 pounds, centrally applied, and a deflection not less than .08 of an inch.

IV. TEST-PIECES AND METHOD OF TESTING.—The standard test shall be 1¼ inches in diameter, about 14 inches long, cast on end in dry sand. The drillings for analysis shall be taken from this test-piece, but in case of rejection the manufacturer shall have option of analyzing drillings from the bore of the cylinder, upon which analysis the acceptance or rejection of the cylinder shall be based.

One test-piece for each cylinder shall be required.

V. CHARACTER OF CASTINGS.—Castings shall be smooth, well cleaned, free from blowholes, shrinkage cracks or other defects, and must finish to blue-print size.

Each cylinder shall have cast on each side of saddle manufacturer's mark, serial number, date made and mark showing order number.

VI. INSPECTOR—The Inspector representing the purchaser shall have all reasonable facilities afforded to him by the manufacturer to satisfy himself that the finished material is furnished in accordance with these specifications. All tests and inspections shall be made at the place of the manufacturer.

Cast-Iron Car Wheels.

The wheels furnished under this specification must be made from the best materials, and in accordance with the best foundry methods. The following pattern analysis is given for information, as representing the chemistry of a good cast-iron wheel. Successful wheels, varying in some of the constituents quite considerably from the figures given, may be made: Total carbon, 3.50 per cent.; graphitic carbon, 2.90 per cent.; combined carbon, .60 per cent.; silicon, .70 per cent.; ·nese, 0.40 per cent.; phosphorus, .50 per cent.; sulphur, .08 per cent.

I. Wheels will be inspected and tested at the place of manufacture.

II. All wheels must conform in general design and in measurements to drawings, which will be furnished, and any departure from the standard drawing must be by special permission in writing, and manufacturers wishing to deviate from the standard dimensions must submit duplicate drawings showing the proposed changes, which must be approved.

III. The following table gives data as to weight and tests of various kinds of wheels for different kinds of cars and service:

Wheel.....	33-Inch Diameter Freight and Passenger Cars.			36-Inch Diameter.	
Kind of service.....	60,000 lb. Capacity and Less.	70,000 lb. Capacity.	100,000 lb. Capacity.	Passenger Cars.	Locomotive Tenders.
Number.....	1	2	3	4	5
Weight. { Maximum....	590 lb.	650 lb.	720 lb.	705 lb.	760 lb.
Weight. { Minimum.....	560 lb.	610 lb.	670 lb.	680 lb.	720 lb.
Height of drop, ft.....	12	12	12	12	12
Number of blows.....	10	12	15	12	15

IV. Each wheel must have plainly cast on the outside plate the name of the maker and place of manufacture. Each wheel must also have cast on the inside double plate the date of casting and a serial foundry number. The manufacturer must also provide for the guarantee mark, if so required by the contract. No wheel bearing a duplicate number, or a number which has once been passed upon, will be considered. Numbers of wheels once rejected will remain unfilled. No wheel bearing an indistinct number or date, or any evidence of an altered or defaced number, will be considered.

V. All wheels offered for inspection must have been measured with a standard tape measure and must have the shrinkage number stenciled in plain figures on the inside of the wheel. The standard tape measure must correspond in form and construction to the "Wheel Circumference Measure" established by the Master Car Builders' Association in 1900. The nomenclature of that measure need not, however, be followed, it being sufficient if the graduating marks indicating tape sizes are one-eighth of an inch apart. Any convenient method of showing the shrinkage or stencil number may be employed. Experience shows that standard tape measures elongate a little with use, and it is essential to have them frequently compared and rectified. When ready for inspection, the wheels must be arranged in rows according to shrinkage numbers, all wheels of the same date being grouped together. Wheels bearing dates more than thirty days prior to the date of inspection will not be accepted for test, except by permission. For any single inspection and test, only wheels having three consecutive shrinkage or stencil numbers will be considered. The manufacturer will, of course, decide what three shrinkage or stencil numbers he will submit in any given lot of 103 wheels offered, and the same three shrinkage or stencil numbers need not be offered each time.

VI. The body of the wheels must be smooth and free from slag and blowholes, and the hubs must be solid. Wheels will not be rejected because of drawing around the center core. The tread and throat of the wheels must be smooth, free from deep and irregular wrinkles, slag, sand wash, chill cracks or swollen rims, and be free from any evidence of hollow rims, and the throat and thread must be practically free from sweat.

VII. Wheels tested must show soft, clean, gray iron, free from defects, such as holes containing slag or dirt more than one-quarter of an inch in diameter, or clusters of such holes, honey-combing of iron in the hub, white iron in the plates or hub, or clear white iron around the anchors of chaplets at a greater distance than one-half of an inch in any direction. The depth of the clear white iron must not exceed seven-eighths of an inch at the throat and one inch at the middle of the tread, nor must it be less than three-eighths of an inch at the throat or any part of the tread. The blending of the white iron with the gray iron behind must be without any distinct line of demarcation, and the iron must not have a mottled appearance in any part of the wheel at a greater distance than one and five-eighths inches from the tread or throat. The depth of chill will be determined by inspection of the three test-wheels described below, all test-wheels being broken for this purpose, if necessary. If one only of the three test-wheels fails in limits of chill, all the lot under test of the same shrinkage or stencil number will be rejected, and the test will be regarded as finished so far as this lot of 103 wheels is concerned. The manufacturer may, however, offer the wheels of the other two shrinkage or stencil numbers, provided they are acceptable in other respects as constituents of another 103 wheels for a subsequent test. If two of the three test-wheels fail in limits of chill, the wheels in the lot of 103 of the same shrinkage or stencil number as these two wheels will be rejected, and, as before, the test will be regarded as finished so far as this lot of 103 wheels is concerned. The manufacturer may, however, offer the wheels of the third shrinkage or stencil number, provided they are acceptable in other respects, as constituents of another 103 wheels for a subsequent test. If all three test-wheels fail in limits of chill, of course the whole hundred will be rejected.

VIII. The manufacturer must notify when he is ready to ship not less than 100 wheels; must await the arrival of the Inspector; must have a car, or cars, ready to be loaded with the wheels, and must furnish facilities and labor to enable the Inspector to inspect, test, load and ship the wheels promptly. Wheels offered for inspection must not be covered with any substance which will hide defects.

IX. A hundred or more wheels being ready for test, the Inspector will make a list of the wheel numbers, at the same time examining each wheel for defects. Any wheels which fail to conform to specifications by reason of defects must be laid aside, and such wheels will not be accepted for shipment. As individual wheels are rejected, others of the proper shrinkage, or stencil number, may be offered to keep the number good.

X. The Inspector will retape not less than 10 per cent. of the wheels offered for test, and if he finds any showing wrong tape marking, he will tape the whole lot and require them to be restenciled, at the same time having the old stencil marks obliterated. He will weigh and make check measurements of at least 10 per cent. of the wheels offered for test, and if any of these wheels fail to conform to the specification, he will weigh and measure the whole lot, refusing to accept for shipment any wheels which fail in these respects.

XI. Experience indicates that wheels with higher shrinkage or lower stencil numbers are more apt to fail on thermal test; more apt to fail on drop-test, and more apt to exceed the maximum allowable chill than those with higher stencil or lower shrinkage numbers; while, on the other hand, wheels with higher stencil or lower shrinkage numbers are more apt to be deficient in chill. For each 103 wheels apparently acceptable, the Inspector will select three wheels for test—one from each of the three shrinkage or stencil numbers offered. One of these wheels chosen for this purpose by the Inspector must be tested by drop-test as follows: The wheel must be placed flange downward in an anvil block weighing not less than

1,700 pounds, set on rubble masonry two feet deep and having three supports not more than five inches wide for the flange of the wheel to rest on. It must be struck centrally upon the hub by a weight of 140 pounds, falling from a height as shown in the table in Section III. The end of the falling weight must be flat, so as to strike fairly on the hub, and when by wear the bottom of the weight assumes a round or conical form, it must be replaced. The machine for making this test is shown on drawings which will be furnished. Should the wheels stand, without breaking in two or more pieces, the number of blows shown in the above table, the one hundred wheels represented by it will be considered satisfactory as to this test. Should it fail, the whole hundred will be rejected.

XII. The other two test wheels must be tested as follows: The wheels must be laid flange down in the sand, and a channelway one and one-half inches in width at the center of the tread and four inches deep must be molded with green sand around the wheel. The clean tread of the wheel must form one side of this channelway, and the clean flange must form as much of the bottom as its width will cover. The channelway must then be filled to the top from one ladle with molten cast-iron, which must be poured directly into the channelway without previous cooling or stirring, and this iron must be so hot, when poured, that the ring which is formed when the metal is cold shall be solid or free from wrinkles or layers. Iron at this temperature will usually cut a hole at the point of impact with the flange. In order to avoid spitting during the pouring, the tread and inside of the flange during the thermal test should be covered with a coat of shellac; wheels which are wet or which have been exposed to snow or frost may be warmed sufficiently to dry them or remove the frost before testing, but under no circumstances must the thermal test be applied to a wheel that in any part feels warm to the hand. The time when pouring ceases must be noted, and two minutes later an examination of the wheel under test must be made. If the wheel is found broken in pieces, or if any crack in the plates extends through or into the tread, the test-wheel will be regarded as having failed. If both wheels stand, the whole hundred will be accepted as to this test. If both fail, the whole hundred will be rejected. If one only of the thermal test wheels fails, all of the lot under test of the same shrinkage or stencil number will be rejected, and the test will be regarded as finished, so far as this lot of wheels is concerned. The manufacturer may, however, offer the wheels of the other two shrinkage or stencil numbers, provided they are acceptable in other respects, as constituents of another 103 wheels for a subsequent test.

XIII. All wheels which pass inspection and test will be regarded as accepted, and may be either shipped or stored for future shipment, as arranged. It is desired that shipments should be, as far as possible, in lots of 100 wheels. In all cases the Inspector must witness the shipment, and he must give, in his report, the numbers of all wheels inspected and the disposition made of them.

XIV. Individual wheels will be considered to have failed and will not be accepted or further considered, which,

1. Do not conform to standard design and measurement.
2. Are under or over weight.
3. Have the physical defects described in Section VI.

XV. Each 103 wheels submitted for test will be considered to have failed and will not be accepted or considered further, if,

1. The test-wheels do not conform to Section VII., especially as to limits of white iron in the throat and tread and around chaplets.
2. One of the test-wheels does not stand the drop test as described in Section XI.
3. Both of the two test-wheels do not stand the thermal test as described in Section XII.

Malleable Castings.

I. PROCESS OF MANUFACTURE.—Malleable iron castings may be made by the open hearth, air furnace or cupola process. Cupola iron, however, is not recommended for heavy or for important castings.

II. CHEMICAL PROPERTIES.—Castings for which physical requirements are specified shall not contain over .06 sulphur nor over .225 phosphorus.

III. PHYSICAL PROPERTIES.—1. *Standard Test-Bar.* This bar shall be 1 inch square and 14 inches long, without chills and with ends left perfectly free in the mold. Three shall be cast in one mold, heavy risers insuring sound bars. Where the full heat goes into castings which are subject to specification, one mold shall be poured two minutes after tapping into the first ladle, and another mold from the last iron of the heat. Molds shall be suitably stamped to insure identification of the bars, the bars being annealed with the castings. Where only a partial heat is required for the work in hand, one mold should be cast from the first ladle used and another after the required iron has been tapped.

2. Of the three test-bars from the two molds required for each heat, one shall be tested for tensile strength and elongation, the other for transverse strength and deflection. The other remaining bar is reserved for either the transverse or tensile test, in case of the failure of the two other bars to come up to requirements. The halves of the bars broken transversely may also be used for tensile strength.

3. Failure to reach the required limit for the tensile strength with elongation, as also the transverse strength with deflection, on the part of at least one test rejects the castings from that heat.

4. *Tensile Test.* The tensile strength of a standard test-bar for castings under specification shall not be less than 42,000 pounds per square inch. The elongation measured in 2 inches shall not be less than 2½ per cent.

5. *Transverse Test.* The transverse strength of a standard test-bar, on supports 12 inches apart, pressure being applied at center, shall not be less than 3,000 pounds, deflection being at least ½ of an inch.

IV. TEST LUGS.—Castings of special design or of special importance may be provided with suitable test lugs at the option of the Inspector. At least one of these lugs shall be left on the castings for his inspection upon his request therefor.

V. ANNEALING.—1. Malleable castings shall neither be "over" nor "under" annealed. They must have received their full heat in the oven at least sixty hours after reaching that temperature.

2. The "saggers" shall not be dumped until the contents shall at least be "black hot."

FINISH.—Castings shall be true to pattern, free from blemishes, scale or shrinkage cracks. A variation of $\frac{1}{16}$ of an inch per foot shall be permissible. Founders shall not be held responsible for defects due to irregular cross sections or unevenly distributed metal.

INSPECTION.—The Inspector representing the purchaser shall have all reasonable facilities given him by the founder to satify him that the finished material is furnished in accordance with these specifications. All tests and inspections shall be made prior to shipment.

Gray Iron Castings.

I. PROCESS OF MANUFACTURE.—Unless furnace iron is specified, all gray castings are understood to be made by the cupola process.

II. CHEMICAL PROPERTIES.—The sulphur-contents to be as follows:

Light castings, ...	not over 0.08 per cent.
Medium castings, ...	not over 0.10 per cent.
Heavy castings, ...	not over 0.12 per cent.

III. DEFINITION.—In dividing castings into light, medium and heavy classes, the following standards have been adopted:

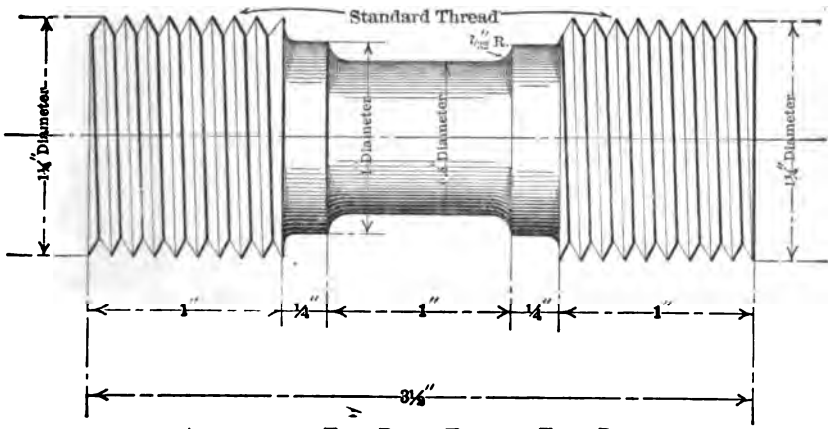
Castings having any section less than ½ of an inch thick shall be known as light castings.

Castings in which no section is less than 2 inches thick shall be known as heavy castings.

Medium castings are those not included in the above definitions.

IV. PHYSICAL PROPERTIES.—*Transverse Test.* The minimum breaking strength of the "Arbitration Bar" under transverse load shall be not under: Light castings, 2,500 lb.; medium castings, 2,900 lb.; and heavy castings, 3,300 lb. In no case shall the deflection be under 0.10 of an inch.

FIG. 1.



ARBITRATION TEST-BAR. TENSILE TEST PIECE.

Tensile Test. Where specified, this shall not run less than: Light castings, 18,000 lb.; medium castings, 21,000 lb.; and heavy castings, 24,000 lb. per sq. in.

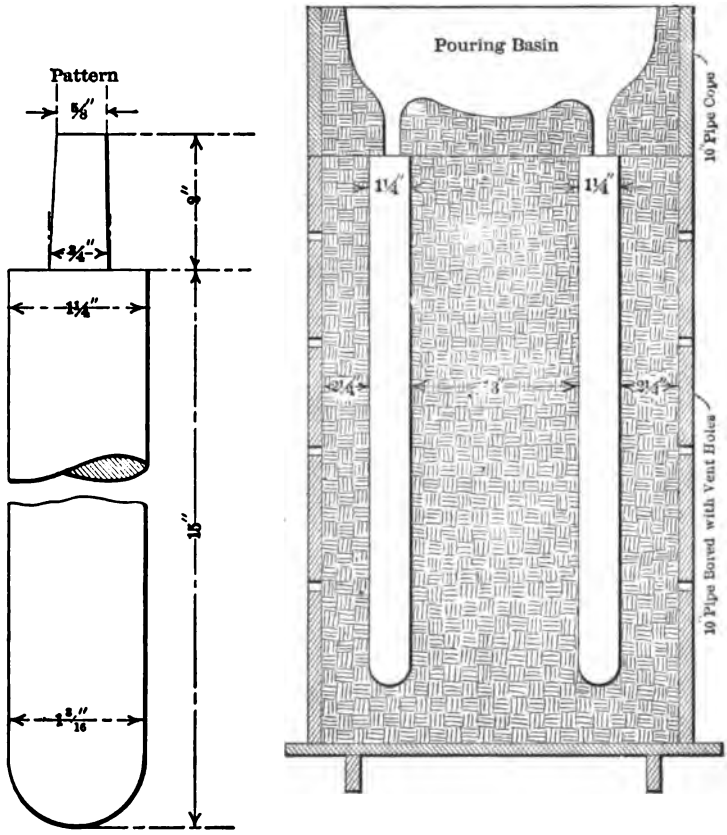
V. THE "ARBITRATION BAR" AND METHODS OF TESTING.—The quality of the iron going into castings under specification shall be determined by means of the "Arbitration Bar." This is a bar 1¼ inches in diameter and 15 inches long. It shall be prepared as stated further on and tested transversely. The tensile test is not recommended, but in case it is called for, the bar, as shown in Fig. 1, and turned up from any of the broken pieces of the transverse test, shall be used. The expense of the tensile test shall fall on the purchaser.

Two sets of two bars shall be cast from each heat, one set from the first and the other set from the last iron going into the castings. Where the heat exceeds twenty tons, an additional set of two bars shall be cast for each twenty tons or fraction thereof above this amount. In case of a change of mixture during the heat, one set of two bars shall also be cast for every mixture other than the regular one. Each set of two bars is to go into a single mold. The bars shall not be rumbled or otherwise treated, being simply brushed off before testing.

The transverse test shall be made on all the bars cast, with supports 12 inches apart, load applied at the middle, and the deflection at rupture noted. One bar of every two of each set made must fulfill the requirements to permit acceptance of the castings represented.

The mold for the bars is shown in Fig. 2. The bottom of the bar is $\frac{1}{16}$ of an inch smaller in diameter than the top, to allow for draft and for the strain of pouring. The pattern shall not be rapped before withdrawing. The flask is to

FIG. 2.



MOLD FOR ARBITRATION TEST-BAR.

SECRETARY'S NOTE.—These illustrations have not been drawn to scale.—R. W. R.

be rammed up with green molding sand, a little damper than usual, well mixed and put through a No. 8 sieve, with a mixture of one to twelve bituminous facing. The mold shall be rammed evenly and fairly hard, thoroughly dried and not cast until it is cold. The test-bar shall not be removed from the mold until cold enough to be handled.

VI. SPEED OF TESTING.—The rate of application of the load shall be thirty seconds for a deflection of .10 of an inch.

VII. SAMPLES FOR CHEMICAL ANALYSIS.—Borings from the broken pieces of the "Arbitration Bar" shall be used for the sulphur determinations. One determination for each mold made shall be required. In case of dispute, the standards of the American Foundrymen's Association shall be used for comparison.

FINISH.—Castings shall be true to pattern, free from cracks, flaws and excessive shrinkage. In other respects they shall conform to whatever points may be specially agreed upon.

INSPECTION.—The Inspector shall have reasonable facilities afforded him by the manufacturer to satisfy him that the finished material is furnished in accordance with these specifications. All tests and inspections shall, as far as possible, be made at the place of manufacture prior to shipment.

Chemical Specifications for Pig-Iron.

BY EDGAR S. COOK, POTTSTOWN, PA.

(Atlantic City Meeting, February, 1904.)

PORTIONS of this paper repeat in substance the statements made by me in an address before the meeting of the American Society for Testing Materials, held in June, 1903, at Delaware Water Gap, Pa. The views I then expressed, I still hold and believe to be pertinent to the present discussion of this subject. In doing my part, therefore, at the request of our Secretary, towards the further consideration by the Institute of the Physics of Cast-Iron from a commercial as well as a theoretical standpoint,—a theme which received interesting treatment in our meeting some years ago, but has been somewhat neglected recently,—I make no apology for reiterating the views I have expressed elsewhere.

Indications all point to the substitution of classifying pig-iron by chemical analysis for the old-time method of grading by fracture. Commencing with the larger users, and the makers of specialties, who require more positive information as to the composition than is afforded by the fracture; the use of pig-iron, guided by analysis, has spread with much greater rapidity to all classes of foundry-work than the most sanguine expected.

Grading by fracture, as guided by experience with any particular brand of iron, has some points in its favor. The fact that, for so many years, castings of multitudinous forms, and for an endless variety of uses, were made in a fairly satisfactory

manner by skilful guessing as to mixtures, shows that, for many classes of work, a considerable range in composition may be allowed, without injuriously affecting the castings. But with accurate knowledge as to requirements, and more positive information as to the composition of pig-iron, there will doubtless be a smaller percentage of defective castings.

Working by analysis, however, will not prove a panacea for all the ills of foundry practice. There are many causes affecting the success of such practice other than the composition of the iron charged into the cupola.

There are limitations, also, to the ability of the blast-furnace to furnish pig-iron according to rigid chemical specifications without unduly enhancing the cost. In the long run, the consumer must pay the cost and risks of manufacture, plus a fair profit on the capital invested; otherwise the supply will not equal the demand.

The manufacture of special irons having a certain composition, with very narrow limits of permissible variation, demands special ores and special fuels, as well as uniform conditions of furnace-work. Such ores and fuels are not in abundant supply, and therefore command comparatively high prices; while the unavoidable "misfits" made by the furnace, and sold at less than cost, enhance the average cost of manufacture, for all of which the consumer must ultimately pay.

Blast-furnaces could be run with much more comfort to the manager, and with much better average results in every way, if it were possible to obtain absolutely uniform raw material, and if the atmospheric air driven into the tuyeres always contained the same weights of oxygen and moisture per cubic foot. But if all the furnaces operating to-day were forced to demand ores of uniform composition, always the same in percentage of iron and gangue, and coke that did not vary in sulpur, phosphorus, ash and fixed carbon, most of them would be obliged to blow out; and the cost of pig-iron would rise to prohibitory figures.

The furnaces are obliged, therefore, to take raw materials as they find them, and, by judicious mixing, based upon an approximate knowledge of their contents, make as uniform a product as possible.

This product is the raw material of the foundry, rolling-mill,

steel-works, etc. It is a wise precaution, therefore, in substituting grading "by analysis" for grading "by fracture," not to draw the lines closer, or make the conditions more severe than is absolutely necessary to meet the requirements of the particular class of manufacture concerned.

I have seen crude specifications for pig-iron, probably formulated by a young chemist, fresh from school, but without metallurgical practice, which were so pedantically exacting with regard to carbon, silicon, sulphur, phosphorus, manganese, etc., that probably not one cast in a thousand from any one furnace would meet all conditions imposed. Such specifications indicate ignorance of the inevitable variations in the composition of pig-iron, especially of sand-cast pig, made in the usual way. Different portions of the same pig, different beds of the same cast, and different pigs of the same cast, frequently show considerable variations in sulphur and silicon. Without attempting here to discuss the causes of such variations, we must accept the fact as an unavoidable feature of the conditions existing at most blast-furnaces.

In this connection I cannot do better than quote the remarks of Dr. R. W. Raymond, in a discussion upon "The Control of Silicon in Pig-Iron."[1]

"This apparatus [the blast-furnace] is noted as perhaps at once the rudest and the most sensitive of the means employed in manufacturing on a large scale. In proportion to its rudeness and sensitiveness, it has always called for exceptional skill in management. By skill, I mean here the acquired aptitude of actual practice, including the recognition and the treatment of symptoms, without the perfect knowledge of their causes. The requirement of skill, in this sense, has been greatly diminished by the work of chemists and engineers, during the last few years. Many of the subtle difficulties and traditional remedies of the blast-furnace practice of twenty years ago are practically obsolete in the art of to-day, with its rapid running, high pressures and temperatures, immense daily products and adequate machinery. But the hampering condition still remains, that these great furnaces, devouring hundreds of tons of raw material and producing hundreds of tons of pig-iron and slag daily, are required to preserve a certain chemical composition of product. This requirement is often inconsistent with the use of the cheapest ores, the maximum economy of fuel and the maximum product of iron. In other words, if our blast-furnace managers were ordered to produce the largest practicable quantity of pig-iron, no matter of what grade or quality, at the smallest cost of ore, flux, fuel, labor and repairs, the art would be not only simplified, but revolutionized.

"Now this is what has happened in other arts, as the result of scientific improvement. The production of the cruder material has been immensely cheap-

[1] *Trans.*, xxi., 361 (February, 1892).

ened, by reducing the element of skill, removing to a great extent the requirement of complicated quality, and leaving to subsequent processes, in which science, rather than skill, plays the chief part, the final work of perfection and adaptation.

"The barbaric method (as illustrated, for instance, in the Japanese manufacture of steel) consists in the infinite expenditure, according to received tradition, of material, time and labor; the rejection of the greater part of the product; and the selection of that part which, by happy accident, has acquired extraordinary excellence.

"The half-civilized method substitutes more and more skill for luck, and science for skill, yet still waits to see what the product will be, and 'grades' it, without reference to what it was intended to be.

"The scientific method makes what it started to make. To do this, it requires to know and to govern all the conditions of manufacture, including the exact composition of materials. If this knowledge and control cannot be easily maintained in the first step of manufacture, or if their exercise interferes with the economical operation of that stage on a large scale, the scientific method employs that part of its process as a means of simplifying the conditions for future determination and control."

Blast-furnace practice may be said to be in the "half-civilized" state, we blast-furnace managers having preceded our foundry friends in emerging from the "barbaric" stage by nearly a score of years. But the foundry is really most interested in knowing the exact composition of its raw material—pig-iron. This is more important than having it rigidly conform to any particular set of specifications, defining the limits of silicon, sulphur, etc. It is for the foundry-manager, knowing the composition of his raw material, to employ the scientific method, and, if unable to obtain one pig-iron, which will meet all his requirements, to make such a mixture of various irons as will produce the result he seeks.

The correct sampling of sand-cast pig, especially of the foundry-grades, is troublesome and expensive. For a fairly representative sample, numerous pigs must be drilled with care. Even then, different samplings of the same lot of iron will show frequently, under the same method of analysis, widely varying results as to silicon and sulphur. This occasions misunderstandings, quite as difficult to adjust as disagreements with respect to the "fracture" indicating any particular "grade."

I do not refer only to controversies affecting the price, where judgment may be warped by money-considerations. I have in mind several experiences of our own, illustrating the difficulty of arriving at the composition of sand-cast pig, when the question of price did not enter into the matter at all, as no particular

guarantees had been made. The purchaser required reasonably accurate information to guide him in using the iron; and we co-operated with him in sampling and making analyses. All the samples were made by drilling about one dozen pigs to represent each lot of 50 tons. The comparative results were as follows:

TABLE I.—*Comparative Analyses of Samples of Pig-Iron, Taken by Drilling of Sand-Cast Pigs.*

PURCHASERS.		WARWICK IRON & STEEL CO.	
Silicon.	Sulphur.	Silicon.	Sulphur.
Per Cent.	Per Cent.	Per Cent.	Per Cent.
1.90	trace	0.964	0.045
1.23	0.04	0.926	0.045
0.85	0.09	0.855	0.050
1.71	0.035	0.880	0.040
1.08	0.040	0.910	0.060
1.03	0.06	1.010	0.080
1.04	0.05	1.08	0.050
1.31	0.04	1.19	0.04
1.50	0.03	1.09	0.06
0.62	0.05	0.74	0.04
0.87	0.03	0.80	0.04
1.21	0.04	0.92	0.07
1.04	0.05	1.08	0.05

The marked discrepancies in silicon were found to be due to particles of sand included in the samples, through lack of care in the drilling of the sand-coated pigs in the machine-shop of the purchaser. The iron sold was all "gray forge."

This experience led us to adopt the method of sampling iron liquid, as it ran from the furnace, which was in use by the western furnaces connected with steel-works. It has been found more reliable, as fairly representing iron to be used in quantity, than the drilling of individual pigs.

Sometimes, consecutive casts of pig-iron prove to be remarkably uniform in silicon- and sulphur-content, the variation between the different beds not exceeding the permissible laboratory-variation; but this is the exception rather than the rule. Phosphorus is generally evenly distributed. Segregation does not seem to affect it, and there is seldom any trouble in determining it. Table II., on page 180, illustrates the variation of silicon and sulphur in sand-cast iron, one pig being selected from the middle of every alternate bed.

TABLE II.—*Analyses of Drillings from Pigs of the Same Cast.*

	A. Open Gray Forge.		B. No. 2 Strong.		C. Soft Foundry.		D.	
Bed No.	Silicon.	Sulphur.	Silicon.	Sulphur.	Silicon.	Sulphur.	Silicon.	Sulphur.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
2.....	1.481	0.068	1.669	0.013	2.52	0.011	2.52	0.014
4.....	1.434	0.066	1.575	0.009	2.60	0.015	2.56	0.013
6.....	1.468	0.074	1.599	0.009	2.66	0.013	2.58	0.013
8.....	1.481	0.066	1.599	0.003	2.80	0.013	2.53	0.012
10.....	1.458	0.056	1.622	0.009	2.92	0.013	2.55	0.012
12.....	1.411	0.060	1.575	0.011	3.01	0.011	2.56	0.013
14.....	1.292	0.066	1.528	0.011	2.92	0.011	2.48	0.013
16.....	1.198	0.056	1.528	0.006	2.04	0.014	2.84	0.013
18.....	1.222	0.055	1.411	0.011	.....	.....	.....	.....
20.....	1.222	0.048	1.363	0.008	.....	.....	.....	.....
CAST-ANALYSIS, LIQUID SAMPLES.								
.....	1.363	0.054	1.599	0.012	2.60	0.011	2.50	0.013
.....	.....	.....	.....	.....	2.23	0.013	2.88	0.015

Cast A.—The silicon varies between 1.481 and 1.222 per cent., and the sulphur between 0.074 and 0.048 per cent.

Cast B.—The silicon varies between 1.669 and 1.363 per cent., while the sulphur happens to be exceptionally low.

Cast C.—Shows a wide variation in silicon from 3.01 to 2.04 per cent., while the sulphur is low and constant.

Cast D.—Is uniform in both silicon and sulphur.

Another series might be still more variable in silicon and sulphur, according to furnace-conditions.

Table III. gives the analyses of drillings from different portions of the same cast of sand-cast pig.

TABLE III.—*Analyses of Drillings from Pigs Taken at Random from the Same Cast.*

Grade.	FIRST SAMPLE.		SECOND SAMPLE.	
	Silicon.	Sulphur.	Silicon.	Sulphur.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
No. 2X.....	2.23	0.017	2.45	0.015
No. 2X.....	3.29	0.014	2.30	0.091
No. 2X.....	2.16	0.022	1.46	0.018
No. 2X.....	2.15	0.036	2.40	0.027
No. 2X.....	3.24	0.035	2.16	0.060
No. 2X.....	3.20	0.026	3.32	0.017
No. 2X.....	3.21	0.051	3.17	0.036
No. 2.....	1.62	0.075	2.58	0.033
No. 2.....	1.88	0.035	2.66	0.013
No. 3.....	1.25	0.035	0.89	0.050

Table IV. gives another example of variation in pig-iron analyses.

TABLE IV.—*Analyses of Liquid and Solid Samples from the Same Cast.*

LIQUID SAMPLES.					
	Silicon.	Sulphur.			
	Per Cent.	Per Cent.			
	2.20	0.021			
	2.85	0.017			
DRILLINGS.			DRILLINGS.		
Bed. No.	Silicon.	Sulphur.	Bed. No.	Silicon.	Sulphur.
	Per Cent.	Per Cent.		Per Cent.	Per Cent.
1	1.99	0.026	11	2.60	0.023
		0.022	13	2.66	0.022
3	2.17	0.022	15	2.85	0.020
5	2.29	0.024	17	2.36	0.019
7	2.46	0.024	19	1.80	0.021
9	2.57	0.027			0.024

This cast shows considerable variation in silicon, while the sulphur is uniform.

At what point iron represented by one analysis ends and another begins is difficult to determine. Occasionally, physical indications serve as a guide to the experienced grader.

These illustrations show the difficulties besetting the accurate determination of the composition of sand-cast pig. When the situation is complicated by commercial considerations, it is evident that grading by analysis may not remove all friction of interests, though it is unquestionably an improvement upon grading by fracture, and certainly a safer guide for the use of pig-iron. Fortunately, for most classes of foundry-work, moderate variations in the composition in carefully-selected foundry-irons are not fatal, because a certain range in chemical composition has been found permissible in practice.

At the Delaware Water Gap meeting of the American Society for Testing Materials, the Sub-Committee on Pig-Iron presented a report on the classification of pig-iron, which, after prolonged discussion, was provisionally modified by the action of that meeting. In its modified form, the report was subsequently submitted by letter to the members of the Pig-Iron Sub-Committee, approved by them, and then transmitted to the Secretary of the Society, for further action by the members

thereof. It is not necessary for me to discuss in detail all the differences between it and the original report of the sub-committee. One of them, however, is worthy of notice, as throwing some light upon the problem involved and the spirit in which it has been approached. Namely, the sub-committee originally suggested the provision that "in the absence of specifications" the current grades of pig-iron should be considered as representing the following chemical composition:

Grade.	Silicon. Per Cent.	Sulphur. Per Cent.
"Scotch,"	Over 3.0	Not over 0.03
"No. 1,"	2.5 to 3.0	Not over 0.03
"No. 2X,"	2.0 to 2.5	Not over 0.045
"No. 2,"	1.75 to 2.25	Not over 0.055
"No. 3,"	1.50 and over	Not over 0.075

The modified report omits the grades known as "Scotch" and "No. 2X," and substitutes the following classification:

Grade.	Silicon. Per Cent.	Sulphur. Per Cent.
"No. 1,"	2.75	0.035
"No. 2,"	2.25	0.045
"No. 3,"	1.75	0.055
"No. 4,"	1.25	0.065

with a permissible variation of 0.01 per cent. of sulphur and 10 per cent. of the quantity of silicon. It will be understood that these figures are offered simply as a means of characterizing "grades" of iron, when no chemical specification has been made. Of course, an intelligent and reasonable chemical specification would supersede this provisional classification.

Specifications for Pig-Iron and Iron Castings.

BY ROBERT JOB, READING, PA.

(Atlantic City Meeting, February, 1904.)

UP to five years ago the pig-iron used by the Philadelphia & Reading Railway Co. had been obtained solely upon the appearance of the fracture; but as the service was unsatisfactory, an investigation was made to determine the quality best adapted

to the requirements of the company. It was found in the first place that the proportions of phosphorus and of silicon were high, which resulted in the castings having a rather low tensile strength and being weak under impact. After a study of the quality and conditions of the scrap of the company, as well as of the service desired, specifications were drawn up substantially in the form of those herewith appended, limiting the proportion of phosphorus to 0.5 per cent., and the proportion of silicon to about 1.5 per cent., holding the latter component in practice to about 1.75 per cent. Since some of the scrap was rather high in sulphur, a minimum of 0.4 per cent. was placed on the manganese-content of the pig-iron, and ferro-manganese was used in the ladle. The result of these changes was to produce a tough, close-grained, easily-machined casting, of high resistance under impact.

At the end of the first year after beginning to use this iron, the breakages had decreased to such an extent that the company's supply of scrap was largely cut off; and at the end of the second year the supply had decreased to so low a point that it became necessary to get the scrap for the cupola-furnaces elsewhere, and to purify it by an additional treatment.

Specifications for Iron Castings (Philadelphia & Reading Railway Co.[1]).

1. *Physical Requirements.*

All castings must be of uniform quality, and of solid iron free from physical defects and excessive shrinkage strains, finished in a workmanlike manner, free from sand, and in close accordance with drawings. Castings purchased under Class 1 or Class 2 must be of gray-iron throughout and easily machined.

2. *Chemical Requirements.*

Class 1. *Medium Iron.*—Engine cylinders, gears, wheel centers, smoke stacks, etc. The iron must be close grained and tough. The composition must be silicon, from 1.4 to 2.; sulphur, not exceeding 0.085; manganese, not exceeding 0.7; phosphorus, not exceeding 0.6 per cent.

Class 2. *Soft Iron.*—Small castings for general car and roadway use. The composition must be silicon, from 2. to 2.8; sulphur, not exceeding 0.085; manganese, not exceeding 0.7; phosphorus, not exceeding 0.6 per cent.

Class 3.—Brake-shoes and other castings for frictional wear. The iron must be hard and tough. The composition must be silicon, from 2. to 2.5; sulphur, not

[1] Office of First Vice-President, Philadelphia, Pa., February 24, 1903, Theodore Voorhees, First Vice-President.

exceeding 0.15; manganese, not exceeding 0.7; phosphorus, not exceeding 0.7 per cent.

3. *Method of Inspection.*

Upon receipt of a shipment, a thorough inspection will be made, and only those castings will be considered which meet the requirements of Section 1. From such castings borings will be taken from at least one in each fifty or fraction thereof, and the composition must be within the stated limits or the shipment will be rejected.

4. *Rejected Material.*

All rejected material will be returned at the expense of the shipper, and all castings which fail in service owing to defects of manufacture must be replaced free of cost.

SPECIFICATIONS FOR PIG-IRON—SUPERSEDING PREVIOUS SPECIFICATIONS. (PHILADELPHIA & READING RAILWAY CO.[2])

1. *Physical Requirements.*

Shipments must be of uniform quality, and free from sand, dirt, slag, or other foreign matter. Pigs must be broken, or be of such size as to be easily handled.

2. *Method of Sampling.*

Upon inspection, three pigs will be selected at random from each carload or fraction thereof, and a sample of drillings obtained by boring into the face of a fractured end of each of the three with a blunt, wide-angled two-inch drill, to a depth of not less than two inches, care being taken to have the drillings uniformly fine and free from sand or other foreign matter. The samples from these three pigs will then be thoroughly mixed, and this final sample upon analysis must conform to the following composition, or the carload represented will be rejected.

3. *Chemical Composition.*

The chemical composition must be silicon, from 1.5 to 2.5; manganese, from 0.4 to 0.75; phosphorus, not exceeding 0.50; sulphur, not exceeding 0.04 per cent.

4. *Rejection.*

In case of rejection at the shops, the material will be returned at the expense of the shipper.

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[2] Office of First Vice-President, Philadelphia, Pa., February 24, 1903, Theodore Voorhees, First Vice-President.

Specifications for Cast-Iron and Finished Castings.

BY RICHARD MOLDENKE, NEW YORK CITY.

(Atlantic City Meeting, February, 1904.)

AMONG the things that will always remain to the credit of the foundry-industry is the circumstance that nearly all the research-work of practical value in daily routine was done by active foundrymen, and the results were freely given by them to the world. Moreover, while consumers felt that to them was due a more systematic and exact method of work, the foundrymen again were the first to act in the establishment of specifications under which their product should be sold.

To come to more recent times, the American Foundrymen's Association has not only made an elaborate series of tests on cast-iron, and adopted tentative specifications based upon them, but, in addition, it has urged the International Association for Testing Materials, through its American representatives, to make the testing and specification of cast-iron a distinct issue. The half-hearted support first granted was greatly augmented by the rapidly-growing recognition accorded to the enormous iron-casting industry; and when the president of the American Society for Testing Materials was instructed to ask for a special commission for this purpose, Europe was ready to cooperate. During the Buda-Pesth congress, at which America was represented by Prof. Howe, Mr. Wood and myself, it was decided to add five members to the general committee on steel and iron, whose special duty should be to conserve the interests of the cast-iron branch of the industry.

America took up the work at once, and enlarged its committee to 67 members, all of whom were actively concerned in the manufacture or study of cast-iron in all of its branches. At the organization-meeting in Philadelphia, April 25, 1903, Mr. Walter Wood was elected chairman, and myself secretary; and, within a week thereafter, the sub-committees were appointed and became active at once. The scope of the work was naturally limited to those branches of the iron-casting in-

dustry which lend themselves to standardization; and the following committees began to collect information on existing methods, and to formulate specifications based thereon, as far as possible—the final results being intended to represent the best American practice at the present time.

A sub-committee was appointed for each of the important branches of the cast-iron industry, namely:—pig-iron, pipe, cylinders, car-wheels, malleable cast-iron, general castings, and the testing of cast-iron for quality as a metal. Subsequently, two additional committees were appointed, one on the influence of additions to cast-iron, and the other on the microstructure of cast-iron.

The membership of these committees embraces the consumer, the producer, and the engineer, so that all three interests are represented. Through the spirit of fair play, and the broad-minded view of the situation to which it led, the result has been most satisfactory. The American Society for Testing Materials is steadily becoming the great clearing-house of the nation in matters relating to industrial practice, and the benefits of its work will be incalculable.

The specifications for pig-iron were completed first, and bear the marks of the very recent radical change from the old hap-hazard fracture-specification, to one based upon scientific principles. The trade-customs had to be retained to some extent—otherwise the departure would have been unintelligible to many of our older foundrymen, whom it is proposed to help as much as possible in getting good material. These older foundrymen would be lost if they had to draw from machine-cast pig-iron only, while the modern operator wants all the metal he can get in that form.

The malleable cast-iron specifications were completed next. The testing of cast-iron itself, and the large class of general castings had to be combined—the American view being that if a casting cannot be tested to destruction, the next best thing to do is to get a measure of the quality of the iron in it, under conditions which are both fair and accurate.

Of these questions, further information is given by the statements presented at this meeting of the Institute by the chairmen of the respective sub-committees.

Standard Specifications for Cast-Iron Pipe.*

BY WALTER WOOD, PHILADELPHIA, PA.

(Atlantic City Meeting, February, 1904.)

THE specifications for cast-iron pipe that have been submitted at this meeting are practically the outgrowth of those which were originally adopted, about 1860, by Mr. Kirkwood of Brooklyn, N. Y. They have been changed but little since that time, the mode of testing pipe in the hydraulic press, as well as the method of coating, not having departed from the original practice.

The chief addition to the specifications has been in the adoption of a test-bar 2 in. by 1 in. in cross-section, broken between 24-in. supports by a transverse strain applied on the flat side. Test-bars were first introduced into pipe-specifications about 1870. At that time a satisfactory test-piece had to stand a tensile strain of 16,000 lb. per square inch.

The present test-bar was adopted in order to give some idea of the deflection of the iron that was being used. These deflection-tests have been so satisfactory to both engineers and manufacturers, and have met the requirements of the pipe-trade so accurately, that it is considered best to hold to the 2-in. by 1-in. bar instead of adopting the short "Arbitration Bar," which yields but little information from the stand-point of deflection.

The new feature in these specifications is the effort to reach standards to which pipe shall be cast in the future, thus permitting the manufacturer to make up stock and the purchaser to find material on hand when he asks for deliveries. Heretofore there has never been any standard weight for pipes, although in some sizes certain weights are more frequently asked for than others.

The fixing of standards for pipes permits the adoption of definite standards for fittings. It has been aimed to secure such uniformity for fittings as will permit the same class of fittings to cover two or more classes of pipe. In order to accomplish

* See page 162.

this, the specifications provide that pipe of less than 30 in. diameter shall be made to standard outside diameters; the variation of weights between the classes to be obtained by changing the size of the core. The details governing these adjustments can best be understood by a careful reading of the corresponding specifications.

In closing, it may be well to add that the various details of these specifications have been reached through several extended conferences between engineers using water-pipe, and pipe-manufacturers, and that they are aimed to cover the requirements of both the user and the maker.

It is to be regretted that one point of difference could not be adjusted, namely,—the desirability of casting the smaller pipe with spigot-end down, so that this portion of the pipe should be certain to contain the closest and densest metal. At our conferences, the engineers insisted that the spigot-end should be cast upward in the smaller sizes. How far this point will be adjusted at future conferences between manufacturers and users, remains to be seen.

Standard Specifications for Locomotive-Cylinders.*

BY WALTER WOOD, PHILADELPHIA, PA.

(Atlantic City Meeting, February, 1904.)

THE specifications for locomotive-cylinders, which are given in a separate paper at this meeting, have been prepared so that castings (upon the successful use of which so much depends) shall be made to conform to standards that will yield good results.

After the essentials, which go with a sound and perfect casting, the requirements to be borne in mind are those concerning the quality of the material. These have been determined from the actual practice of the largest users,—representatives of the American Locomotive Company, the Baldwin shops, and the Pennsylvania Railroad being upon the Committee.

The results called for by the test-bar have been carefully checked by actual experiment upon metal that has given satisfactory service in cylinders.

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* See page 168.

The analysis also has been determined by long experience as one that will yield a strong iron and will give a dense and hard wearing-surface.

It has been possible to make the specifications short and simple, and yet to cover the essential points definitely and with thoroughness.

Standard Specifications for Cast-Iron Car-Wheels.*

BY CHARLES B. DUDLEY, ALTOONA, PA.

(Atlantic City Meeting, February, 1904.)

It is evident that, as the size and weight of cars have increased, the demands on the cast-iron car-wheel have become more and more severe. Fortunately, the factor of safety in the cast-iron wheel, as originally made, was so large that it was only when freight-cars approached the capacity of from 80,000 to 100,000 lb. that the question of wheels began to give some anxiety. A moment's thought will show that, as freight- and passenger-cars have increased in weight and size, a much greater duty is required of the same eight wheels than was formerly the case. In other words, although the cars have increased in capacity from 15 tons to 50 tons each, the number of wheels under a car has not been generally increased. It is true, some cars have been equipped with 12 wheels, but this construction is not the rule.

Hence the duty which the wheel must perform has become much more severe, and although it was clearly able to stand some increase, there must evidently be a limit somewhere which cannot be exceeded. Meanwhile, it is the part of wisdom to make better wheels if possible, and accordingly, no little study is now being devoted to the requirements for cast-iron car-wheels; and modifications are being made, both by increasing the weight of the wheel and by improving the quality of the metal of which it is made.

The desire to improve the quality of the wheel has led to the preparation of specifications for its manufacture; and within the last 15 years a more rigid scrutiny of the wheel is char-

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* See page 168.

acteristic of the purchaser, as well as a corresponding effort on the part of the manufacturer to meet special requirements.

The specifications for cast-iron car-wheels, submitted by the sub-committee of the American Society for Testing Materials, are believed to embody the best available knowledge and judgment on the subject, although of course there are some differences of opinion upon single points; and the section devoted to the chemical composition of the cast-iron is given simply for guidance rather than for any special requirement. Concerning these specifications, I may be permitted to offer some further explanations.

Sections I and II are self-explanatory.

Section III gives the proposed weights of wheels for various services, a question concerning which there is a considerable difference of opinion. The manufacturers in general prefer heavier wheels than those given in the specifications. It is obvious that the smaller the weight, the better must be the quality of the metal in order to stand the same strain; and it is natural, in view of the uncertainties of wheel-manufacture, that makers should desire to increase the weight as much as possible, for the reason that metal not quite as good, if there is more of it and it is properly disposed, will give the requisite strength. It is fair to say that the proposed specification is still to be discussed before the American Society for Testing Materials, and it is possible that the weights therein given will be modified.

Section IV needs no special remark.

Section V is devoted to the subject of tape-sizes. No foundry is able to make its total output all of the same circumferential size, and experience has shown that there is an intimate relation in any good foundry between tape-sizes and satisfactory wheels.

There are several reasons why the output of a foundry varies in diameter or circumferential measurement. First, although the molds are supposed to be of the same size originally, as a matter of fact they are not. Moreover, a mold which has been used a number of times is apt to have been increased a little in diameter, as well as to have become worn a little. This cause of variation in tape-sizes is not very serious, because efforts are usually made to keep the molds fairly uniform in size.

Another cause of variation in size, is the temperature of pouring the metal; and it will be readily understood that greater shrinkage is characteristic of metal cast at higher temperatures. Furthermore, a difference in chemical composition makes a difference in the shrinkage. However, the most important cause for variation in tape-sizes is the effect of the annealing. It is well known that as fast as the wheels are taken out of the molds they are put, while they are still red-hot, in annealing-pits containing 15 wheels or more. The annealing-pits are constructed of metal tubes lined with fire-brick; the interstices between the pits are filled with sand; and the whole is arranged so that the wheel may cool slowly. Generally, the wheels remain in the pits for four days, during which time certain changes take place in the metal, and in most cases the tape-size of the wheel is increased. Experiment has indicated that a re-annealing, that is, putting a wheel into the pit a second time among a number of freshly-cast hot wheels, will at times increase the tape-size two numbers. Furthermore, the wheels at the top and bottom of the pit do not increase in size in the same proportion as do those in the middle of the pit.

It is perhaps not necessary at this time to go into the changes which take place in the annealing-pit, further than to say that those wheels which come out of the pit nearest to the size which they had when put into the pit, or in other words, those wheels which are less annealed, are found by experience to be most likely to fail on the drop-test, and are also less likely to stand the thermal test. It will be seen, therefore, that there is a very intimate relation between tape-sizes and satisfactory product—which explains why so much reliance is placed on the tape-sizes. The wheel-circumference measure used by the Master Car Builders' Association is a brass tape about an inch wide, having supports at different points, so that the wheel may be measured on the tread at a uniform distance from the flange.

In the wheel-making business, the manner in which tape-sizes shall be specified is not uniform. The Master Car Builders' Association has a nomenclature prescribing "0" for a standard-size wheel, plus 1, 2 and 3, for wheels larger than the standard, and minus 1, 2, 3, for those smaller than the standard.

Some works have an arbitrary nomenclature,—for instance, the Altoona wheel-foundry standard size is 120, but wheels of 119, 121, 122, 123, etc., are put in service. As stated in Section V, the nomenclature is immaterial, it being essential only that the different sizes shall differ from each other by $\frac{1}{8}$ inch.

The reason of the clause which requires that wheels not more than 30 days old shall be submitted for the test, is that the date of the guaranty is taken from the date cast in the wheel, and if wheels remain two or three months before test, the date of the guaranty will be unfair.

The reason for considering only three sizes at one time will be discussed later in this paper.

Section VI, referring especially to the physical appearance, will be readily understood by any one familiar with foundry-work in the manufacture of wheels.

Section VII deals with the chill. It is well known that a cast-iron car-wheel is practically a gray-iron wheel having a tire, if it may be so called, of chilled iron. The chilled iron of the tire is brittle, extremely hard, and not of much value, so far as strength is concerned. It is the gray iron that is relied upon for strength. Of course it is understood that the chilled iron is a part of the casting, the chill being formed by having that part of the mold, which is to produce the chill, made of metal, the rest of the mold being made of sand. In casting the wheels, if the iron is of the proper chemical composition, that part of it which touches the iron of the mold is instantly cooled and becomes white chilled iron, while the part which touches the sand in the mold becomes gray iron. As already explained, the tape-size is an important element in the wheel, and in addition to strength and ability to stand the thermal test, the chill likewise is a function of the tape-size. The lowest tape-size in any foundry will be apt to have the highest chill, and may fail on test from having too much chill, while the higher tape-sizes, which mean a greater circumference, have the lesser chill and may fail from having too little chill.

The requirements of Section VII, referring to rejections, are based on wide experience. It was formerly the custom to reject an entire lot of 100 wheels, if any of the test-wheels failed from any cause. This custom was considered a hardship by the manufacturers; and in view of the intimate relation between

tape-sizes and successful wheels, there has been introduced into these specifications, the authority to reject, out of each 100 wheels tested, only the wheels of the same tape-size as the wheel which failed, thus allowing the other tape-sizes to come up again. This has not been tried, so far as is known; but those who have had most experience in wheel-manufacture and have done most work in testing the wheels, see no reason why any greater risks would be incurred by the user in following this plan, than under the old plan of rejecting a hundred wheels for the failure of one. At the same time, it enables the manufacturer to produce a greater proportion of satisfactory wheels.

Sections VIII, IX and X are self-explanatory, and need no special remarks.

Section XI concerns the drop-test. It will be noted that what has been said about tape-sizes permeates the whole specification. This is especially true with regard to the drop-test and the thermal test. According to the specification as proposed, each tape-size gets a test, either a drop or a thermal one. It was formerly the custom to allow four, five or possibly six tape-sizes in a hundred wheels, and to allow the inspector to choose the wheels for test arbitrarily. It is believed that the testing of each tape-size is a decided step forward in this matter; and if this practice is followed, not only will there be greater certainty of getting good wheels, but also, as the specification is drawn, less hardship will be put upon the manufacturer.

It will be observed that only one of the three wheels is subjected to the drop-test. There is much difference of opinion among wheel-makers and consumers concerning the real value of this test. Many consumers do not hesitate to say that they think it has very little value, since it is so easy to strengthen the wheel at those points where experience has shown them to fail under the drop-test, that, with any kind of metal, wheels can be made which will stand it. A very slight thickening of the plates without any change in the quality of the metal, together with a slight diminution in thickness at other points where the wheel has never failed, so as not to exceed maximum weights, will enable a foundry to turn out wheels that will stand the drop-test. However, there are believed to be some advantages in testing under the drop;

and many consumers have been unwilling to abolish it entirely. Those who do not attach importance to it rely almost wholly on the thermal test, and think that everything covered by the drop-test can be determined as well, or better, by the thermal test. But car-wheels in actual service do receive considerable shocks; and it is some satisfaction to know that the wheels will stand them without breaking; hence it seems probable that the drop-test will maintain a place in specifications for some time to come.

Section XII is devoted to the thermal test, which is believed to be by far the most important, and the one which tells the most about the wheels. An account of the origin of this test may be interesting. Until recently, the larger part of the failures of cast-iron wheels were found to occur at, or a little distance beyond, the end of a long-continued application of the brakes, as, for example, in going down a long grade. This naturally led to a study of the reasons of the bursting, cracking or breaking of wheels under cars at these points. A little reasoning on the matter, together with an investigation of the wheels after they had reached the bottom of the grade, soon showed that the application of the brakes resulted in a heating of the outside circumference of the wheel. Strangely enough, it was possible to find in a wheel under a car which had had the brakes applied down a long grade, that the rim or outer portion of the wheel was frequently hot enough to burn the fingers, while toward the hub it was apparently absolutely cold. In other words, the heat generated by the friction of the brake did not transmit itself throughout the whole wheel uniformly, until considerable time had elapsed. Not infrequently, it takes half an hour for a train to run down a long grade, and during this time, the metal six inches away from the rim of the wheel is quite cold to the hand, while the rim itself is quite hot. The heating of the rim of course produces expansion; and this causes a good deal of strain between the metal at the rim and that at the center of the wheel, a result which frequently causes the bursting and failure of the wheel in service. The thermal test is designed to simulate the conditions of a long-continued application of the brake. It will be seen from the descriptive matter in Section XII that the rim of the wheel is highly

heated by the molten cast-iron poured against it, while the center is not heated, thus reproducing the conditions caused by a long-continued application of the brakes, perhaps more severe than would ever occur in service.

During the development of the thermal test many wheels of various makes were tested; some burst within a few seconds after the pouring of the hot metal was finished, so much so that, at times, the molten metal was spattered around by the bursting. Some stood for a longer period of time, while others stood until the molten metal had become black-hot, or even cooled down to ordinary temperatures. It was also found that the wheels cracked in eight different ways. Some of these breaks or cracks, such, for instance, as the breaking of the brackets, were considered harmless, and only two of the eight forms were embodied in the test, namely, breaking in pieces, and cracks involving the tread.

As already mentioned, the length of time from the cessation of pouring until the cracks appeared was found to vary from a few seconds to about seven minutes, and in order to make the specification workable, a period of two minutes was chosen as the time through which the wheel must be tested. It is believed that no test for cast-iron wheels has been suggested that is so important, or that tells so much, or that has done so much to improve their quality, as the thermal test.

There is a difference of opinion concerning the severity of the test. Some think that an inch of molten metal in the channel-way is enough; others that a shorter time-limit should be required. It is evident that, by varying either the thickness of the molten metal, or the length of time, the test may be made more or less severe. Another requirement might be introduced, namely, that no cracks of any kind should be formed. Hence the test taken as a whole may comprise many important possibilities. I have seen many wheels that did not break in any place during the thermal test; but their percentage is small. The provision of Section V, regarding the reappearance of tape-sizes which have not failed, is also made in Section XII as far as the thermal tests are concerned.

Sections XIII, XIV and XV do not require any special comment.

The query may arise, why, if the test-wheel representing a

tape-size passes any one of the tests (such as the chill- or thermal test), these wheels should not be accepted, for the test which they pass successfully, and not be required to be re-tested again as a constituent of a new lot of 103 wheels, according to the specifications. At first sight this seems unduly severe, but it should be remembered that wheels are usually bought in large numbers, that a hundred wheels make a car-load and consequently this number is a convenient unit to be considered, also, that at some of the smaller wheel-foundries (and indeed at some of the larger ones), when the demand for wheels is pressing, often only a hundred wheels are ready for test at one time. If, now, the lot fails from any cause, the inspector must leave this foundry, and wait until another lot is ready. It was felt therefore, that to require the inspector to keep track of partial tests would unduly complicate the inspection, and throw doubt on its value. Moreover, as has already been stated, previous specifications have usually rejected the whole hundred wheels, if any one of the three test-wheels failed on any of the tests; while the present specifications reject absolutely only those wheels of the same tape-size, as the test-wheel which fails, allowing the other tape-sizes to appear for test again, which is as great a modification as it seems wise to make in the present state of making car-wheels. Furthermore, it should not be forgotten that the prime object of the specifications is to get as great certainty as possible, that only safe wheels shall go into the service, and that doubtful ones shall not pass. Finally, since the consumer pays for the test-wheels at a given contract-price, it was felt that no serious hardship is introduced by the proposed specification, and that it is better to err, if at all, on the side of safety.

Taken as a whole, it is perhaps safe to say that, for ordinary service (by which is meant for all service except under 40- and 50-ton freight-cars), the wheels which pass the tests of these specifications will be safe and will give fairly good results. There are some points in connection with the failure of wheels under heavy cars, that need further study; and it is more than probable that, as this study progresses, it may be found essential to change some of the requirements of the proposed specifications. The special failure of wheels under heavy cars is a circumferential crack either in the tread or in the throat

of the wheel, resulting sometimes in the breaking-off of the flange. The causes leading up to this failure are complicated, and it is possible that modifications in the design of the cars themselves may very greatly diminish the number of failures of this kind. On the other hand, a most earnest study of the wheel itself is now being made, in order to enable it to resist successfully the strains producing these circumferential cracks. It is fair to say that there are some who believe that steel-tired wheels or some kind of a steel wheel will be necessary for heavy service. There are also some who are strongly of the opinion that a cast-iron wheel can be made which will successfully meet the conditions under heavy cars, and will thus perpetuate the remarkable and wonderful reputation which the cast-iron wheel has made for itself in this country.

The Need of Standard Specifications for Gray-Iron Castings.*

BY HENRY SOUTHER, HARTFORD, CONN.

(Atlantic City Meeting, February, 1904.)

IT is generally admitted that a good, practical and commercial set of specifications for the use of the many purchasers of cast-iron does not exist. A few good sets are in the hands of large buyers, but even these are very different, contradictory, and not intended to cover a wide range of product. Furthermore, most of them have been prepared to cover but one branch of the industry.

It is strange that purchasers of cast-iron have not looked into this question long ago and demanded good, uniform cast-iron.

Probably cast-iron has been discarded from many uses because of its failure in a few instances, even though the failure was due to ignorance of the quality of the iron, or perhaps to abnormally bad iron. Some foundrymen seem to consider all gray iron from a given foundry as uniform, especially if they know that it has been made from certain well-known brands of pig. It is likely that with standard specifications and the increase of knowledge necessary for their intelligent employ-

* See page 172, *et seq.*

ment, the use of gray-iron castings will enter into fresh fields because of the resultant certainty of their quality. This was the case when steel replaced wrought-iron, for the reason that steel was made of uniformly good quality and was thoroughly reliable. The foundryman, therefore, should welcome any commercially practicable step in the direction of securing standard specifications for cast-iron.

Uniformity of product in soft steel is comparatively easy to bring about, and yet the most rigid specifications are in common use by the purchasers of this material. Uniformity of product in cast-iron is not easy to attain because of the greater quantity of impurities in it; yet specifications for soft steel are common while for cast-iron they are rare.

In steel, only carbon, phosphorus and sulphur are of prime importance (manganese, silicon and copper being secondary), and all these impurities exist in very small percentages.

In cast-iron, however, silicon, sulphur, carbon and phosphorus are present in comparatively large percentages and are of vital importance. Furthermore, each impurity is capable of notably altering the quality of the metal. Manganese also, as a secondary element, should be intelligently regulated. Yet cast-iron varying widely in composition is neither watched, inspected nor tested, while soft steel, which may vary but little in comparison, is submitted to careful examination before it is accepted. This circumstance is not due to the fact that cast-iron is not used at critical points; because columns are made of it, the breaking of which might wreck a building. Moreover, cast-iron enters into the manufacture of high-pressure valves and fittings essential to the safety of a boiler or a hydraulic system; it has also many other important uses.

Not only is cast-iron subject to chemical and physical differences under normal conditions, but, like most cast metals, it is liable to have bad internal flaws and latent defects not detectable by any other means than a destructive test. In many cases these defects are the indirect results of bad chemical conditions; for example, a high sulphur percentage, and for this reason if for no other, a limit to the sulphur-content should have a place in the specifications for its manufacture.

I. Process of Manufacture.

In certain grades of gray iron unusual strength is demanded, and in many instances it is convenient to make this grade in a reverberatory or in a cupola. Under the specifications submitted, either method is allowed, both being believed to be good; but, unless otherwise specified, the cupola will be used. Some founders are successful in making all grades of castings in the cupola; but in this case both iron and fuel must be used with intelligence.

II. Chemical Properties.

It is believed that light castings should contain less sulphur than heavy ones, because sulphur induces chill, shrinkage and such flaws as are grouped under the broad head of "dirty iron." All these defects are more serious in light work than in heavy: the chill, because it is possible in thin iron and not in thick iron, the latter containing so much heat as to cause it to anneal itself in cooling; shrinkage, because light castings chill quickly, form a small, close grain and shrink more in consequence, whereas heavy castings cool slowly, form large, open grain and shrink much less; and flaws, because a flaw of a given size forms a much more serious defect in a light casting than in a heavy one. Flaws induced by the presence of sulphur are not proportionate in size to the bulk of the casting; and it is common to find shot-holes in small castings of about the same size as those which exist in much larger ones.

The sulphur-limits fixed in these specifications are within the commercial reach of any well-regulated foundry, and necessitate no extra cost. The sulphur-limit for heavy castings has been objected to as being unnecessarily low; but it is on the safe side, is an insurance against bad flaws, and therefore should be retained, especially since it does not increase the cost of the iron.

Cast-iron with high sulphur-content is sometimes stronger than with low, because the grain in the former instance is closer—a condition which always tends to increase the strength; but of what use is the extra strength if, at some vital point in the casting, flaws exist as a result of high sulphur, which weaken it and far more than offset the extra strength of the sound parts of the casting. Even if a test-bar representing a

cylinder does show unusual strength and is of high sulphur-content and sound at the point of rupture, the casting itself may contain enough shot- and blow-holes to cause its failure.

High sulphur is a menace to any casting, whether it can be detected physically or not; and this impurity should be kept as low as is commercially possible.

III. Definition.

The classification made is approximate and will naturally be supplemented by buyers who will specify that a given casting must conform to the specifications applying to one of the grades. This arrangement removes all uncertainty concerning the grade of iron intended.

IV. Physical Properties.

The figures given in Table I. have been taken from as many tests as was possible to accumulate in the short time available after having determined the dimensions of the test-specimen. All kinds of iron are not represented, but those that are given show very encouraging results.

I was not able to get reliable information in all cases. The bars are from six foundries that make products of very varied character.

From the same foundries the results of testing the inch-square bars selected at random from recent tests, are given in Table II. for comparison, the tests having been made under exactly similar conditions. Tests Nos. 12,761 and 12,762 of Table II. were of metal from the same ladle as Tests Nos. 12,759 and 12,760 of Table I.

The theoretical ratio of the strength of the inch-square bar in common use as compared with the arbitration-bar is about 0.8 to 1, or, to be exact, 0.867; that is, the inch-square bar is about 80 per cent. as strong as the 1.25-in. round bar, all things being equal. The results of the few tests made in both shapes from the same run is close to this, as is shown in Tables I. and II. Actual results do not follow the theory exactly, for the reason that the greater volume of the arbitration-bar results in its being softer, all other things being equal. There is also a lesser proportion of chilled metal in the round bar.

TABLE I.—*Results Obtained from Tests Made with Arbitration Test-Bars.*

No. of Specimen.	Deflection in 12 Inches.	Strength.	Phosphorus.	Manganese.	Sulphur.	Silicon.
	Inches.	Pounds Central Load.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
12,174	0.15	3,500	0.630	0.30	0.128	1.819
12,175	0.12	3,600				
12,266	0.09	3,500 (Cast with fins.)	0.889	0.60	0.144	2.383
12,267	0.07	3,200				
12,269	0.07	3,200	0.653	0.18	0.104	2.040
12,270	0.08	3,200				
12,343	0.13	3,300	0.520	0.36	0.099	1.683
12,344	0.11	3,000				
12,345	0.12	3,200	(These bars had fins.)			1.584
12,346	0.13	3,000				
12,429	0.10	2,800	0.534	0.36	0.098	1.725
12,430	0.08	2,800				
12,431	0.10	3,500	(All cast with fins.)			1.810
12,432	0.10	3,100				
12,433	0.13	3,500	0.340	0.30	0.113	1.340
12,434	0.11	3,800				
12,435	0.11	3,500	0.364	0.28	0.101	1.678
12,436	0.12	3,400				
12,480	0.14	3,600	0.681	0.30	0.105	2.092
12,481	0.12	3,800				
12,546	0.07	3,200	0.562	0.32	0.095	1.899
12,547	0.11	3,400				1.848
12,551	0.09	2,700	0.326	0.38	0.088	1.565
12,552	0.09	3,000				
12,553	0.06	2,600	0.377	0.36	0.097	1.631
12,554	0.09	3,100				
12,638	Lost.	3,500	0.536	0.42	0.088	1.974
12,639	0.10	4,100	0.687	0.24	0.089	1.960
12,673	0.10	2,900	0.580	0.34	0.124	1.706
12,674	0.13	3,550				
12,675	0.11	3,500	.....	.....	.....	1.852
12,676	0.11	3,200				
12,729	0.15	3,500	0.758	0.20	0.099	2.101
12,730	0.11	2,800	0.756	0.20	0.111	2.125
12,731	0.15	3,000	0.269	0.30	0.084	1.184
12,732	0.13	3,100				
12,733	0.17	3,500	.....	.....	.....	1.800
12,734	0.15	3,200				
12,752	0.14	2,500	0.631	0.43	0.092	2.453
12,753	0 17	2,500	(These bars had fins.)			
12,759	0.10	4,000	(Semi-steel.) 0.382	0.16	0.170	1.462
12,760	0.10	4,000				
12,780	0.09	2,700	0.716	0.46	0.047	2.468
12,781	0.11	3,200				
12,800	0.13	2,800	0.690	0.48	0.048	2.695
12,801	0.13	2,800				
12,838	0.15	3,700	0.413	0.36	0.089	1.730
12,839	0.15	3,000				
12,840	0.14	3,100	.....	.....	0.094	1.645
12,841	0.16	3,800				
12,863	0.10	3,100	0.723	0.30	0.136	2.275
12,864	0.10	3,100				
12,913	0.12	3,300	0.663	0.32	0.129	1.683
12,914	0.13	3,300				
12,915	0.13	3,200	.....	.....	.....	1.749
12,916	0.08	2,900				

The results bracketed together signify that the iron was from the same day's cast—sometimes out of the same ladle, and sometimes not.

TABLE II.—*Results Obtained from Tests Made with 1-in. Square Bars.*

No. of Specimen.	Cast No.	Strength.	Phosphorus.	Manganese.	Sulphur.	Silicon.
		Pounds Central Load.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
12,006	2	2,700	0.765	0.36	0.072	3.305
12,011	2	2,700	0.605	0.24	0.105	2.718
12,027	11/28	2,800	0.551	0.30	0.118	1.673
12,048	11/20	3,000	0.779	0.46	0.095	2.251
12,065	11/27	3,300	0.722	0.36	....	1.951
12,079	113	2,300	0.595	0.56	0.085	2.016
12,118	2	3,000	0.580	0.16	0.081	2.769
12,157	206	3,000	0.656	0.68	0.082	1.894
12,158	207	3,000	0.651	0.36	0.115	1.866
12,170	12/10	2,300	0.670	0.88	0.074	2.947
12,198	12/5	2,600				1.434
12,200	12/8	2,700	0.535	0.32	0.110	1.640
12,202	12/10	2,800				1.419
12,376	12/19	2,400	0.644	0.70	0.097	3.158
12,420	214	3,000	0.626	0.38	0.085	2.031
12,421	215	2,500	0.614	0.38	0.091	2.007
12,427	2	2,500	0.577	0.24	0.087	2.797
12,761	.....	2,800	Deflection in 12 in.,		0.13 in.	.....
12,762	.....	3,000	Deflection in 12 in.		0.13 in.	.....

The worst enemy of the cast-iron test-specimen is the physical defect or flaw. Fewer of the round test-bars contain flaws than do the square, and it is apparent that the sharp corners, by capillarity or similar influences, attract rising bubbles of gas and entrap them in the quicker-cooling metal contained in the corners. This effect alone renders the round bar superior to the square as a fair representative of a cast.

The figures given in Table I. show that the transverse strength-limits have been placed so low that no difficulty should be experienced in filling the requirements.

After obtaining more data it may be found that higher strength-limits are possible and advisable. With the ones given, however, uniformity and good quality are insured, which is the main object desired at present.

No upper strength-limit has been placed as this factor is controlled by the machining-properties of the iron. It is my opinion that a drilling-test would be the best safeguard against iron too hard. As not enough data exist at present to make proper commercial limits, this matter is one that should be thoroughly investigated in the near future.

A minimum limit for deflection has been placed in the specifications in order to guard against a strong but brittle iron. Such iron is not often met, but nevertheless it is a factor which

should be considered. Some of the tests given show a lower deflection, but most of these were due to the fact that the bars were cast with a sectional mold, the attached fins being subsequently ground off; other low deflections were due to accidental defects, and in one set, the directions for casting had not been followed and the bars had been roughly ground which removed the skin in spots.

The normal specimens cast as directed were better than the standard demanded and were as good as that warranted by the chemical composition.

IV. *a.* Tensile-Strength Test.

Nothing new is offered in this section. The limits of strength correspond to the ordinary demands of the present trade. A uniform gripping-device should be adopted as far as possible for the reason that the grip is of vital importance in testing cast-iron for its tensile strength.

V. Arbitration-Bar.

It was the first choice of the committee that a 1.5-in. bar should be used, in fact, a bar that would be as large as possible in order to be free from all chilling influences and defects resulting from sudden cooling; in short, a less sensitive bar. It was seen that such a bar would break at so great a strength as to be beyond the limits of the majority of testing-machines now in use in foundries. A diameter of 1.25 in. was therefore adopted as generally satisfactory.

It has been suggested that specifying several standard sizes for test-bars, instead of one, would permit the selection of the size most nearly representing the thickness, and therefore the actual cooling-conditions of the casting under inspection. But any bar separately cast would inevitably have its own rate of cooling, which will not be that of the casting; and any coupon, poured as a part of the casting, would have to be machined before testing; and such testing made after the skin has been removed does not represent the actual casting. In short, it is impossible to duplicate conditions in this way; and any method which professes to do so is likely to be more misleading than helpful.

On the other hand it is possible to estimate the strength of any other bar or casting, allowance being made according to

experience for the different conditions, just as the steel-maker rolls a standard test-bar, say 2.5 by $\frac{5}{8}$ in. in section from a small 6 in. square test-ingot, and from the results obtained, is able to foretell the strength of the same steel rolled into a $\frac{5}{8}$-in. rod, a 0.5-in. angle-iron, or a 1.25-in. plate. It is all a question of heat-treatment and speed of cooling, and in this respect cast-iron is similar to steel. Cooling-conditions follow definite laws that are well recognized by the average founder without an analysis of his reasoning. Any grade or quality of cast-iron may be purchased according to the arbitration test-bar as specified, as well as by a dozen different sizes of test-bars. It is a matter of comparison at the best, and one standard is enough.

As a study it would be interesting to cut out specimens from all sizes of castings as far as possible, and utilize the results to form a basis for future comparison, but as a commercial custom this suggestion is practically out of the question.

The importance of the tensile test has been minimized because of the great difficulty of making it truly tensile. Abnormal strains are sure to creep in, except in the most carefully constructed machines operated by the most skilled observers.

The transverse test is within the reach of all; in fact, it corresponds with the conditions of actual use much closer than does the tensile test, and no serious abnormal strains are possible with the ordinary testing-machine.

The occasion is likely to arise in which the tensile test will be advisable, and to meet it, the provision noted in the specifications is inserted. The shape of the specimen resembles quite closely the one in use by the Government, but it has been modified to meet the needs of the piece from which it is to be cut, as well as other practical requirements. The bar has been made as short as possible in order to eliminate the chance of including flaws, yet it is no longer than the Government specimen which is likely to give abnormally high results.

V. *a.* Conditions of Casting.

Probably the most important source of diverse results in cast-iron tests has been the uncertain conditions incident to molding and casting. There never has been any uniformity in this respect, yet it is a well-known fact that cast-iron is very much influenced by various chilling-effects, and is extremely

sensitive to all heat-variations. Some foundrymen have made the molds in sand that is quite moist, and others in sand that is very dry. Some have knocked the test-bar out of the mold as quickly as it became cool enough to stand it. Some have cast the bar on end; some flat; and some inclined, the variations being nearly as many as the number of founders. It has therefore been stipulated that the mold shall be dry and cool before the bar is cast and that the other conditions shall also be observed which will tend to produce uniform castings.

A few of the bars submitted for the test were cast in a parted mold, producing fins of greater or less thickness which were chipped or ground off. The result of this condition was that less deflection was obtained, even though the fins were placed at the neutral axis of the test-specimen.

The provision stating that the test-bars should not be treated in any respect before testing has been found to be most important. The facing of the mold has been particularly specified, so that the test-bar shall strip from the mold in fairly good shape and be ready for the test.

The objection has been made that the bars could not be made sufficiently uniform in diameter—an effect which is more or less true when the iron is cast in a green-sand mold; but with the stated casting-conditions which prohibit the rapping of the pattern, and with the use of a dry mold and a given facing, this objection has become of extremely small importance and has almost disappeared. Measurements made of 20 bars show a maximum variation of but 0.039 in. in an average diameter at the break of 1.236 inches.

TABLE III.—*Measurements of Arbitration Test-Bars at the Point of Rupture.*

No. of Specimen.	Diameter in Inches.	No. of Specimen.	Diameter in Inches.	No. of Specimen.	Diameter in Inches.
1	1.240	8	1.246	15	1.180 This bar cast with fins.
2	1.241	9	1.273	16	1.236 Rounded off by grinding.
3	1.231	10	1.231	17	1.243
4	1.221	11	1.234	18	1.245
5	1.233	12	1.242	19	1.259
6	1.244	13	1.220 This bar was ground smooth.		
7	1.236	14	1.233	20	1.238

It is interesting to note that the only bar cast with fins, that is, in a parted mold, was the most abnormal of the lot, a result which ought to be considered a most excellent endorsement of the proposed method of molding.

The question of the character of the supports to be used during the test has been raised, but it is believed that little need be feared on this account, as the testing-machines in common use are safe in this respect. In the tests given in Tables I. and II., the point of contact where the load was applied has a radius of about 0.25 in., but no mark is left on the casting, and it is doubtful if this would cause any difference unless the edge were exceedingly sharp, or the surface were flat. The two end-bearings used were flat surfaces intended to rotate on a round bearing. The rotation was not perceptible. In all probability, either bearings of this kind, or blunt knife-edges, would answer equally well.

VI. Speed of Testing.

The time-limit of the test has been placed to meet ordinary practice, as near as could be ascertained. In all events it is within reason; and as long as the practice is made uniform by different observers, it is of comparative small moment.

VII. Samples for Chemical Analysis.

This simple provision has been made with the understanding that the boring shall be taken after removing all of the surface-matter. In case graphitic carbon is present, the sample should be taken across the entire face or cross-section of the bar and thoroughly mixed,—a procedure which is necessary because of the difference in the percentage of graphitic carbon between the exterior and interior portions of a specimen of the size in question.

I recognize that these specifications for gray-iron castings are not ideal from every standpoint, nor is it possible that any commercial ones can be. Theory must yield to practice all along the line, and every engineer or manager discovers this fact very early in his experience with conditions of manufacture.

The sub-committee believes that, by means of these specifications, the purchaser of gray-iron castings will be able to obtain

a more uniform and reliable product. It is also certain that there will be correspondingly less friction, because an order for iron-castings will be definite as to quality; and if the foundry makes them to fill the specifications, there can be no dispute concerning the quality of the iron furnished, even though some machinist may think that it machines hard or looks weak.

Notes on Rail-Steel.

BY ROBERT W. HUNT, CHICAGO, ILL.

(Atlantic City Meeting, February, 1904.)

I HAVE repeatedly said that the mechanical treatment of the metal forming a steel rail, during its manufacture, was comparatively of much greater importance than its chemical composition; and years of observation have confirmed and emphasized this fact.

The comparison of the wear of the earlier steel rails with that of later ones, and to the great disadvantage of the latter, is still being constantly made by railway officials. And while some of us, with experience covering the whole history of the manufacture of steel rails, are aware of the great differences in the conditions governing their production at various periods, I do not think the engineering world generally appreciates the direct and imperative influence which these variations have had, and are having, upon the wearing-quality of the rails.

In the earlier days, the steel was poured into ingots which would make but two 30-ft. rails not exceeding a weight of 60 lb. per yard—giving a mass weighing, say about 1,400 lb., and of a section of about 12 in. square. To-day the ingots are about 22 in. square, and weigh more than 4,000 lb. Of course, the interior of the larger ingots must remain hot and liquid longer than that of the smaller ones, and from this condition arises the steel-rail makers *béte noire*—segregation of the metalloids and piping of the steel. Ingots of the smallest section will pipe, but with the increase of the size, so will follow that of the interior cavity. This well-known tendency demands that special care should be exercised to avoid the evils arising from it.

In the earlier days of making steel rails, after the ingot had been rolled down to a bloom of 6 or 7 in. square, all cracks were carefully chipped out of it. Such defects could not be welded up by subsequent working, but if cut out to the deepest point, particularly if the forming of sharp corners was avoided, the steel would, when further rolled, draw from the bottom up; and so if the cavity were not too deep, a sound bar or rail would result. This chipping was performed under a steam-hammer. Later, if while the bar was passing between the rolls of the rail-mill defects were discovered, the rolling-operation was suspended until they could be chipped out. Then, again, great care was taken that the steel bloom should not be overheated. There were from six to eight blooms, each of a size to produce one rail, charged at one time into a heating furnace; and skilled workmen attended to their heating, turning them over on the bottom of the furnace so that all sides should become of an equal temperature. If from any cause this man made a mistake, and sent his steel to the rolls in an unsatisfactorily heated condition, the head rail-roller, or some other mill-official, rejected it, and it was returned to him for further treatment. This meant that more or less care was exercised on each and every rail; but the daily production was, when viewed from to-day's standpoint, quite small. The first departure was to cease chipping the blooms at the rail-rolls—the next, to make it one continuous process from the first blooming of the ingot to the finished rolling of the rails. This procedure stopped the intermediate chipping of the blooms under the steam-hammer, and carried with it the rolling of more than one length rail, that is, the rolling of a mass of steel large enough to produce more than one rail length, and the subsequent sawing of this into two or more rails.

This new method of rolling had been made possible by the introduction of more or less automatic machinery; and the daily production of a given rail-mill increased very rapidly. But, I regret to say, that the care which it was possible to bestow on the making of each individual rail decreased in an even greater ratio,—an effect which was inevitable.

In the old days, a Bessemer converting-house was equipped with two converters, each of about 5 tons' capacity, which was gradually enlarged to 7, 10, 15, and even 20 tons, and addi-

tional converters added. Of course, the size of the house, blowing-engines, cranes, etc., were all proportionately increased and the development of the plant has proceeded until, instead of about 12,000 tons of ingots per month coming out of one converting-house, more than 70,000 tons per month are now produced. There is more and larger machinery, and it has been said that a resultant large output is the best evidence of uniformity in running the mills. That is true so far as mere production goes; but the speed and momentum are against the exercise of the proper kind of care necessary to produce sound ingots of the highest quality of steel. This doctrine is old-fashioned, but it is true. What are we going to do about it? I am not at all certain that I know; but I do think that the condition should not be tacitly accepted, and no effort made toward better things.

Of course, if the same radical course should be pursued with rail-ingots as is done with those intended for ordnance, armor or heavy-shafting purposes, by which the upper portion of each ingot, amounting to quite one-fourth of its entire weight, is cut off and treated as scrap, the greatest danger arising from segregation and piping would thus be eliminated. This procedure will reduce the output of finished rails, and in that and other ways, will add much to the cost of making them; but if it be necessary in order to obtain safe rails, should not the situation be boldly faced? If railways must pay more money, and can by so doing secure rails which will not only be safe against breakage under traffic, but also give better wear, will it not be economy for them to make the greater investment? That would be one way to meet the difficulty, but I believe there are others also.

Alexander L. Holley had more to do with the introduction in America of the Bessemer process for making steel than any other man; and, later, probably did more than any one else to make possible its greatly increased and cheapened output. Some years after that process was firmly established in this country, the Siemens-Martin or open-hearth process was developed, and Mr. Holley was convinced that it also had a great future in America. So outspoken was he in support of this, that on one occasion an intimate associate of his, and one of the "Bessemer Family of Boys," said to him: "It sounds strange for you, of all men, to advocate so strongly the merits

of the open-hearth process." He replied: "You mark my words, the open-hearth process will live to attend the funeral of the Bessemer process in America."

There are a number of the "Boys" still alive, and great changes along the line of Holley's prophetic vision have occurred; so that to them its fulfilment does not seem as improbable as when it was made.

In my judgment, the one question of ore-supply will, in time, compel the increased use in this country of the basic open-hearth process. We cannot forever continue the rejection of ores which are in every other quality suitable for making steel, because their phosphorus-content is outside of the Bessemer limit. And if at the same time, by the use of the other process, we produce a better metal, at cost not much if any greater, the outcome is inevitable.

Phosphorus is the controlling element in rail-steel. If that can be practically disregarded, no one will deny the ability to make a better article, no matter for what purpose it may be intended.

So far as rails are concerned, the theory relating to the difference in the wear of the steel made by the two processes is being subjected to the crucial test of practice. But no matter what steel is used, care must be exercised in making it; in pouring the ingots; in handling and heating them; and in the rolling and straightening of the rails.

While I have at different times prepared and presented to this Institute specifications for the manufacture of rails, I feel that under all the existing conditions the present is not the time for the adoption of any specification as standard. For the last few years, commercial conditions have been such that there has been practically but one side to the market. Such is not now exactly the case. The rails of heavy sections have not been giving the anticipated service. Both Bessemer and basic open-hearth rails are being made in America, and placed in service, side by side. Foreign rails of both basic and acid Bessemer steel have been imported. Several of the large railway systems have contracted for rails under somewhat new specifications. Let us, before saying which, or any, are the best, await results of actual experience. But I repeat, no matter what chemical formula, or what process of making steel, is selected, unless care in manufacture is exercised, all will have been in vain.

Direct-Metal and Cupola-Metal Iron Castings.

BY THOMAS D. WEST, SHARPSVILLE, PA.

(Atlantic City Meeting, February, 1904.)

A SHORT time ago, I had occasion to cast iron plates 1 in. thick, direct from metal containing Si, 0.51; S, 0.045; Mn, 0.75; and P, 0.094 per cent. Much to my astonishment, I found that there was no trouble in planing them; whereas, if cupola-metal of like composition had been used, the planing would have been a difficult operation.

I have often noticed a greater fluidity or "life" in furnace-metal, as compared with the cupola-product. Iron can be seen flowing down the furnace-runners from 50 to 100 ft., and reaching the last pig-bed in satisfactory condition, while cupola-metal might have solidified before it had flowed so far. I have seen Bessemer iron, which, on account of its low phosphorus-content, does not have the "life" of ordinary foundry-iron, held in a 30-ton ladle for nearly an hour and a half; and after skimming off the coke-dust, it still had to be cooled off considerably before being cast.

As a general proposition, it is known that the lower sulphur-content and the frequently higher temperature of direct-metal, compared with the same iron remelted in the cupola, has much to do with its greater life; yet some problems connected with this phenomenon seem to call for further study. For instance, furnace-metal containing more than 1 per cent. of silicon holds carbon, which is separated later as "kish." This separation does not take place to any great degree while the metal is very hot; but during its gradual cooling, at times, the kish is given off in such large quantities as to cover the ground from 20 to 30 ft. around the ladle. In making castings of direct-metal, especially with a high content of silicon, the separation of kish becomes an annoyance.

Perhaps this throwing out of the excess of carbon makes the life of the metal shorter, after it has been remelted in a

cupola. It would be well to ascertain the carbon-content of a ladle of iron during the stages of the cooling-process, and thus to determine the effect of the carbon-reduction more accurately. Concerning hardness, the higher carbon-content undoubtedly has much to do with keeping metal with the lower percentages of silicon softer, than if it were a cupola-product of like composition, excepting the total carbon-content. Perhaps some of our furnace-friends can throw light on this interesting matter.

Notes and Observations on Cast-Iron.

BY J. E. JOHNSON, JR., LONGDALE, VA.

(Atlantic City Meeting, February, 1904.)

THE brief contribution of Mr. West* furnishes a text for the present paper, which will, however, take a wider range, warranted by the writer's somewhat unusual opportunities for the study of the practice of the foundry as well as the blast-furnace, and especially of the production of satisfactory castings from iron of a composition altogether unsuitable, in the opinion of the average foundryman, for that purpose.

Mr. West has observed that plates cast from direct-metal of a certain composition could be planed, whereas this would have been impracticable with cupola-metal of like composition; that furnace-metal has more "life" (*i. e.*, fluidity, or a higher temperature) than cupola-metal; that it throws out much "kish," etc. And he suggests that blast-furnace managers may throw further light on the greater softness of direct-metal castings. Perhaps such light may be furnished to some extent by the present paper.

The difference between cupola-metal and furnace- or direct-metal is, first of all, in the lower average sulphur-content of the latter. It has been proved by experiment that iron increases in its content of sulphur to a considerable extent with each remelting. More than thirty years ago Fairbairn showed that a certain iron after remelting 18 times had only five-

* See page 211.

eighths of its initial tensile strength and stood one-third of its original deflection, though there had been a decided increase in both up to the twelfth remelting. He did not show the cause of these results, but they undoubtedly were due jointly to increase in the content of sulphur and combined carbon.

It is easy to calculate that, if iron be remelted with a coke containing a normal quantity of sulphur, say 1 per cent., with a melting ratio of 10 to 1, and if only half of the sulphur goes into the iron, the increase in the sulphur-content of the latter will be 0.05 per cent., which, according to the usual specifications, is the maximum amount the iron is permitted to contain, as it comes from the furnace.[1]

The reasons for the higher heat of the furnace-metal are: first, that the ascending current of gases has a much lower temperature in the cupola than in the furnace because of the use of cold blast in the former. (This might be overcome in practice by the use of an excessive quantity of fuel, but it would require more than would be commonly supposed); and second and more important, that in the cupola the iron is exposed to the hot gases for a much shorter period, and is consequently unable to approach as closely to the temperature of the gases as it does in the furnace, to say nothing of the temperature being lower in the former than in the latter.

Mr. West's experience in making workable castings from furnace-metal of a silicon-content which, if present in iron cast from the cupola, would put machining out of the question, and his contention that the hotter the metal is poured, the softer the casting, may best be considered together, as they are so closely related.

The experience mentioned can be confirmed without doubt or hesitation. It is, and has been for years, the practice of the works with which I am connected to make, for repairs, as well as for new construction, all castings within the reach of their molding-facilities, directly from a furnace producing "basic" iron, with the silicon guaranteed under 1 per cent. and the sulphur under 0.05 per cent. The iron would perhaps average, the year round, sulphur 0.03 and silicon 0.7 per cent.; and yet there are comparatively few days in the year when the ordinary

[1] In a later portion of this paper will be found a few observations on the mitigation of this evil.

run of castings may not be made without fear of their being too hard to be machined. Of course, when the furnace produces white-iron showing white clear through the pigs (which are cast in chills), it is necessary for the foundry to work on castings which do not require to be machined. On the other hand, should the iron contain much above 0.9 per cent. of silicon, difficulty is experienced in preventing holes and soft places in the castings, caused by the deposition of graphite or "kish"[2] during or after pouring.

The best way to mitigate or prevent this trouble, which is sometimes very annoying, is to pour the iron very hot when making pieces of small or moderate size. The effect is to chill the metal from a high heat by contact with the cold mold and reduce it to the solid form as quickly as possible, thus preventing, to a great extent, the formation of graphite while the iron is molten, which gives rise to the kishy places. The graphite formed after solidification is distributed more or less uniformly throughout the metal; whereas, if the metal is allowed to cool to some extent before pouring, graphite begins to form within it, and separates out at eddies and high points of the mold, with objectionable results.

It is impossible to discuss this subject satisfactorily without making use of the well-established fact that cast-iron being a member of the same series as steel, the principal effects in it are due, as in steel, to the quantity of combined carbon it contains. If the graphite, which is merely enmeshed in the cast-iron, were absent, the latter would be, in fact, an impure steel, containing in general more silicon and (excepting Bessemer pig) more phosphorus than ordinary steel.

This view was urged by me at some length in a paper published about four years ago,[3] and was soon after expounded by Prof. Howe in such a way as to leave no room whatever for doubt.

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[2] "Kish" may be defined as consisting of loose particles of graphite, either thrown off into the air, or gathered on or near the surface of castings in aggregates so dense as wholly or largely to exclude the iron. In this paper, the term graphite, as distinguished from kish, means the particles more or less regularly disseminated and completely enclosed in the iron. In substance, apart from their mode of occurrence, the two are of course identical.

[3] The Chemistry and Physics of Cast-Iron Briefly Considered, *American Machinist*, April 5 and 12, 1900.

The acceptance of this view renders available for the study of cast-iron an enormous amount of material gathered from both practical and theoretical sources through many years, for the solution of problems presented by steel, in fact the work done upon steel furnishes a solid foundation for the study of cast-iron. This, indeed, seems quite the natural order, since it proceeds from the simpler to the more complex compound.

A chart of the "freezing-point curves" of the iron-carbon systems, such as those of Backhuis-Roozeboom and Roberts-Austen, shows that, with a pure cast-iron containing more than about 4.3 per cent. carbon, there is a range of temperature within which carbon must separate out in the form of graphite before solidification takes place; this range of temperature, being closely proportional to the excess of carbon above the quantity mentioned, is roughly 100° F. for each 0.5 per cent. carbon.

If, then, we had to deal with a pure iron containing, say, 5 per cent. of carbon, it is obvious that carbon in considerable quantity would have to separate out of the iron during cooling in the molten condition, and that some of it would naturally pass off into the air, especially in the ordinary operation of casting from a furnace, in which the running of the metal affords every opportunity for the liberation of this graphite. But, as there is little or no iron made containing 5 per cent. of carbon, this explanation alone is insufficient.

The diagram of freezing-point curves referred to shows that, with all pure cast-irons, the separation of graphite continues for several hundred degrees below the point of solidification; and it has been proved by direct experiment that silicon exercises a powerful influence in assisting or compelling the formation of graphite in this temperature-range (a fact also well known in practice); also that it lowers very markedly the temperature at which the separation of graphite can begin, and reduces the quantity of combined carbon with which the iron can be "in equilibrium."[4]

That this action of silicon on the formation of graphite persists above the point of solidification, is proved clearly by its action in relation to suddenly-cooled or chilled castings. It is well known that in the presence of more than 1.5 per cent. of

[4] The Equilibrium of Iron-Carbon Systems, G. Charpy and L. Grenet, *American Manufacturer*, June 5, 1902. *Metallographist*, July, 1902.

silicon it is impossible to obtain any chilling action, and that, other things being equal, the strength of this action is inversely proportional to the quantity of silicon present below that amount.

So much being established, it may be taken as certain that silicon has the same effect in lowering the temperature and the percentage of combined carbon at which the cast-iron is in equilibrium above, as well as below, the point of solidification. It thus lowers the percentage of carbon above which graphite is forced to separate out from the still-molten iron, and this accounts for the observed fact that, at a furnace running on foundry-iron, the cast-house is often full of kish, while at a furnace running on low-silicon iron this is seldom seen, though the carbon-content of the iron is as high (or higher) in the second case as in the first, and the iron may show a beautiful gray fracture when slowly cooled.

Many interesting facts have been developed as a result of using iron, the composition of which is at the extreme limit permissible for foundry-work. Castings are sometimes made from iron, of which it is known that the machining will be, at best, barely practicable; yet it is necessary to save them if possible. For such cases a very slow cooling has proved efficacious; the castings are buried in hot sand and kept there over night, or even for a whole day, if they are large and able to hold their heat for this length of time. Of course, the slow cooling is practically identical with the annealing of tool-steel, with the additional condition of the separation of graphite.

The reason for the absence of kish in ordinary cupola foundry-work is, that the metal, having a temperature considerably lower than that of furnace-metal, has to pass, in cooling, through a much smaller part of that range above the melting-point, within which graphite is compelled to separate out. Consequently, there is little or no formation of graphite during the melted state, and hence no kish. This fact does not prevent the iron from being very graphitic when cold, because, as shown by the freezing-point curve, the separation of graphite continues for several hundred degrees below the temperature of solidification.

It may be added that for each kind of cast-iron there is a maximum of carbon which it can contain dissolved when it "sets," though much of this carbon subsequently passes into graphite in the solid state. Any excess above this limit will

be expelled in the form of graphite while the iron is still liquid. This maximum represents, in fact, the eutectic of iron and carbon for that kind of cast-iron. The limit of carbon for pure cast-iron is 4.3 per cent., as already stated; and this limit diminishes with progressive increase of silicon.

If, for convenience, iron containing more than this amount of carbon be called supercarbonized, or (adopting the scientific term) "hyper-eutectic," and any increase in temperature above the melting-point of the eutectic be called "superfusion," it may be said that kish is formed when a supercarbonized or hyper-eutectic iron is superfused.

It may be said also that kish is formed by molten iron, while graphite is formed by solidified iron; and it must be noted that kish can be formed only when the quantity of silicon present is sufficent to cause the carbon dissolved in the iron to supercarbonize it, or constitute a hyper-eutectic solution. Thus, when first-class basic iron is cast in sand, it may have a most beautiful face and fracture which would cause it to be graded as No. 2 X; yet no kish will be seen flying in the cast-house.

It is impossible to emphasize too strongly the influence of time in all these matters. This is well understood with regard to steel, but seems never to have been thoroughly realized with regard to cast-iron, in which most of the changes common to steel occur, as well as those incident to the formation of graphite.

The diagrams of freezing-point curves before mentioned, which show the cooling-curves and the internal changes of steel, are always based on the condition of slow cooling, therefore do not directly represent the effects of quick cooling.

The effect of quick cooling is to transfer a condition represented in such a diagram by the initial temperature, to the lower point on the same ordinate which corresponds to the final temperature of the quick cooling, thus introducing conditions which the diagram does not completely represent, especially if further cooling, at a slow rate, takes place.

Such a diagram, therefore, does not directly exhibit the results of the chilling effect of the mold on hot-poured metal; yet it does show plainly that denying the time required for the separation of kish from the super-heated metal, increases the carbon-content of the iron at solidification. This seems to me to indicate, further, that quick cooling raises the temperature

of solidification, and consequently increases the range, both of temperature and time, during which graphite is formed after solidification. Hence a hot-poured iron tends to be less kishy and more graphitic than if poured at a lower temperature.

These conclusions are confirmed by practice. Hot-pouring to prevent the formation of kish has been mentioned already, and it may be added that, of castings made from some kinds of furnace-iron, those first poured from the ladle are entirely free from kish, while those poured last from the same ladleful are so kishy as to be objectionable.

So far as I know, these general facts and their fundamental relation to the accepted theory of the general subject have not been published hitherto.

It is worthy of note, in this connection, that hot-pouring tends to prevent the production of hard castings from relatively hard iron, and of kishy castings from relatively soft and kishy iron. This may explain the strong insistence of foundrymen upon "hot" iron.

Concerning the formation of graphite, it must not be forgotten that, as Prof. Howe has pointed out, the important question is not how much graphite is formed, but how much combined carbon is left.

Observations of irons habitually low in both silicon- and sulphur-content make much plainer the action of the latter element than do those of irons further removed from the chilling-point by the presence of more silicon; and it seems to me, as a result of several thousand such observations, that the harmful effects of sulphur have never been properly emphasized.

So far as I know, a lower sulphur-content than 0.05 per cent. is seldom or never specified in foundry or other standard varieties of iron, and yet an iron with 0.05 per cent. of sulphur is no more to be compared with a similar iron containing 0.025 per cent. of sulphur than coke-made white iron is to be compared with cold-blast charcoal-iron.

At the works above mentioned, as at many others, it is the custom to take from every cast a small chill-cast sample (roughly 1.25 in. by 2 in. in cross-section by 7 in. in length), remove it from the chill-box when it has cooled to a dull red, quench it as quickly as possible and break it.

When the furnace is running on basic iron of standard com-

position, these chill-samples will show from the faintest line of white at the lower corners, with 1 per cent. of silicon and, say, 0.02 per cent. of sulphur, to dead-white with lower silicon and higher sulphur. Through a very large part of the range of analysis covering the best basic irons low in both sulphur and silicon, these samples are solid-white, and their variations in fracture, according to the sulphur present, are astonishingly great to be covered by the single title "white." It is not necessary to go into a detailed description of these variations here. I will only say that the low-sulphur irons have a beautiful bright, very markedly crystalline structure (the crystals being long slim pyramids with their bases against the sides and bottom of the chilled surface and pointing toward the center), while the higher-sulphur irons have a structure increasingly flat and non-crystalline; and while the general surface of the fracture with low-sulphur irons is always approximately a plane perpendicular to the length of the sample, this is decreasingly so with higher sulphur, the fracture eventually becoming almost conchoidal. Samples of low-silicon iron with what is considered very low sulphur for foundry-iron (from 0.05 to 0.06 per cent. upward), have become cracked in cooling, and at 0.075 per cent. of sulphur, will frequently fall into many pieces at the lightest tap when cold, though, it should be said, this latter condition is apparently more marked at some times than at others, even though the irons contain identical quantities of sulphur, silicon and other elements, as ordinarily determined.

That sulphur, however, is the principal cause of these variations of physical properties, is sufficiently proved by the fact that with experience it becomes perfectly possible to estimate the sulphur in the iron by the fracture of its chill-samples, and, as a general thing, to come within 0.01 per cent. of the laboratory determination.

With iron approaching 1 per cent. of silicon the effects of sulphur are to some extent masked, but not completely. Its effect in reducing the size of the grain, increasing the depth of chill and blackening the color of the gray part of the fracture is plainly apparent.

That sulphur has an effect in producing a chill (throwing the carbon into the combined form), precisely opposite to that of silicon, is shown very clearly by a study of the chill-samples,

but this is so generally understood that there is no need to elaborate it here; though it cannot be too plainly set forth that silicon is not on that account a complete neutralizer of sulphur.

The exact internal effects of sulphur in steel have been clearly shown by Prof. Arnold[5] and Mr. J. E. Stead,[5] and there is little or no reason to doubt that the action is the same in cast-iron. It may be well to recall here that this action, briefly, is to form a very fusible iron sulphide which remains liquid long after the rest of the iron has solidified, and disseminates itself between the grains of the iron, thus severing or weakening their continuity. It is not to be expected that silicon, for which sulphur has little affinity, should neutralize such an influence, whatever its effect may be in opposing that of sulphur on carbon. This point is of much importance practically, for the reason that much foundry-practice at the present time seems to be founded on the belief that silicon is a neutralizer of sulphur.

Since the quantity of sulphur in iron increases considerably at each melting, it is natural that scrap which has had at least one remelting will contain a higher percentage of sulphur, and therefore will have a more marked tendency to chill, than furnace-iron; and as high-silicon iron will undoubtedly soften white-iron, it is perhaps natural to assume that the addition of the former is all that is necessary to soften scrap. Undoubtedly such an addition will counteract, wholly or in part, the chilling tendency; but it does not follow that this remedy will neutralize all the other effects of sulphur. When it is remembered, moreover, that the action of silicon in preventing the formation of combined carbon is certainly completed at, if not below, 2 per cent., the wisdom of using mixtures, which contain more silicon than this limit as they come from the cupola, is open to question, especially as silicon is a hardener on its own account, and beyond this percentage a weakener also. Silicon, when present in proportion greater than 2 per cent., is also a great dirt-producer in castings. This matter of dirt-production does not receive much attention in general; but I feel confident that many sand-holes in castings are due to silica, produced by the combustion of silicon in the cupola, and remaining sus-

[5] Various papers in *Journal of the Iron and Steel Institute.*

pended in the metal until the latter comes to rest in the mold, and gives the silica a chance to separate.

The cleanness of castings made from low-silicon furnace-metal, as compared with those from ordinary cupola-iron, would surprise those who have not witnessed the difference.

Just how far silicon neutralizes the evil effects of sulphur is one of the most important questions of foundry-practice to-day, and should without doubt be investigated by some one able and competent to do it; preferably under the auspices of some engineering society. I have recently outlined what I believe to be a satisfactory method of investigation,[6] and I hope that something in this direction will be accomplished before long.

Not much can be done to remedy high sulphur in the iron after it has passed through the cupola. The addition of manganese in some form in the ladle is the only method which has been tried to any extent; and it has yielded good results in some cases, though its use is open to various objections of a practical kind, such as the high cost of ferromanganese, which is practically the only form of manganese available for the purpose; chilling of the iron in the ladle, if the ferromanganese is charged cold, or the cost of pre-heating it, if it is charged hot. Nevertheless, there are many cases in which it could be profitably used, care being taken not to add too much and thereby increase the percentage of manganese in the iron to a point where hardness would result.[7]

So far as I know, calcium carbide has never been used for this purpose; but it is well worthy of a trial. It compares favorably with ferromanganese in cost, while its combining power and its affinity for sulphur are much greater; it is an endothermic compound, and therefore its decomposition would impart heat to the bath of metal. Moreover, calcium, being a powerful deoxidizing agent, would tend to remove any oxygen present in the form of iron oxide. The possibility of the presence of iron oxide in a highly carbonized metal like cast-iron

[6] A Suggestion for a Test of the Effect of Silicon in Neutralizing Sulphur in Cast-Iron, *American Machinist*, October 29, 1903.

[7] Since this was written, I have heard of a method, developed in Germany, for removing sulphur by adding manganese-ore to the charge. This absorbs the sulphur and slags it off, very much as if the manganese were in the metallic state, but at less expense. I have no exact details of the process and its results; but it is said to have been successfully used in Germany.

is strongly denied by some; but I am more and more convinced that it is present in white iron.

Much can be done, however, to prevent the absorption of sulphur by the iron in the cupola. The first step is, of course, to use iron and coke low in sulphur, the next is to use an iron containing considerable manganese, say from 0.75 to 1.5 per cent., since this quantity slags off the sulphur and prevents it from entering the iron. In the running of the cupola much can be done by slagging it freely, and there does not seem to be any reason why additions of lime should not be made in such quantity as to neutralize the extreme acidity of the slag. If from 1 to 2 lb. of limestone were charged for every pound of silica in the coke-ash, the slag would be very fusible as compared with blast-furnace slags, but enormously more powerful in the removal of sulphur than one without lime. In a 10-ton heat with a melting ratio of 10 to 1 and coke containing 7 per cent. of silica, there would be 140 lb. of silica in the charge; and 200 lb. of lime added to this would have a marked effect in reducing the sulphur-content of the resultant iron if at all excessive, at an utterly insignificant cost.[8] A still more calcareous slag would be inadvisable, under the present conditions of cupola-practice, because, with the cold blast used, a sufficient temperature would not be attained to melt it. But this brings up the question: Why are not cupolas blown with warm blast?

A calculation of the quantities of heat required and expended in melting iron, or an examination of the top of a cupola in operation, will show that there is plenty of available heat, which could be utilized to save fuel. It has been calculated that from 25 to 40 per cent. of the fuel-consumption could be saved by heating the blast; and this would permit the use of relatively calcareous slags which would reduce the sulphur-content and give better iron—results opposite to those obtained in general practice, when any attempt is made to economize fuel with cold blast.

The means of heating the blast could be very simple, merely

[8] Über die Wirkung des Kalksteinzuschlägs auf das Schmelzgut biem Kupolofenschmelzen, by F. Würst. *Stahl und Eisen*, January 1, 1904. Translated in the *American Machinist*, May 12 and 19, 1904, under the title, The Effect of Limestone Additions on the Product in Cupola Practice. This excellent article describes a very complete series of practical tests and their results.

some wrought- or cast-iron pipes placed in an enlargement of the cupola-stack and connected by return bends, with external joints out of reach of the fire. The quantity of air admitted at the charging-door would give a simple means of controlling the temperature of the gases issuing from the top of the furnace, and thus any damage to the pipes could be prevented. The blast would pass through the pipes in series on its way to the wind-box, and could probably be heated as hot as desired without difficulty and without excessive cost for repairs. The pipes would be exposed to no danger of being burnt up while no air was passing through them, because at such times the cupola would not be going to any extent worthy of consideration.

Many other phenomena concerning this interesting subject have come under my observation, but as these relate to the production of cast-iron (furnace-practice), rather than to its use, I have not included them in the present paper.

The Mobility of Molecules of Cast-Iron.*

BY A. E. OUTERBRIDGE, JR., PHILADELPHIA, PA.

(Atlantic City Meeting, February, 1904.)

I HAD the honor to present to the American Institute of Mining Engineers at the Pittsburg meeting, February, 1896, a paper having the foregoing title, in which I said: "It has generally been accepted as a fact that cast-iron, under the influence of repeated shocks, becomes brittle, and will finally break under a blow which otherwise it would have withstood. It will probably surprise metallurgists, therefore, to learn that experiment disproves the supposed fact, and establishes exactly its opposite.

"The result of about a thousand tests of bars of cast-iron of all grades, from the softest foundry mixtures to the strongest car-wheel metal, enables me to assert with confidence that, within limits, cast-iron is materially strengthened by subjection to repeated shocks or blows."

Three tables were appended showing increase of strength

* Supplementing the Paper under this Title in *Transactions*, xxvi., p. 176.

and resilience of test-bars which had been subjected to repeated shocks or vibrations, in various ways, as compared with companion bars cast in the same molds from the same ladles of iron, not so treated. The largest gain in strength recorded in these tables was very nearly 19 per cent.

The surprising statements immediately attracted the universal notice of founders and engineers, both in this country and in Europe; many tests were made and recorded in technical journals, which not only corroborated the original claims but largely exceeded them.

The Franklin Institute appointed a committee of experts to investigate the subject. This committee cast a large number of bars, half of which were subjected to mechanical shocks, the remaining half being untreated. Their report (No. 1,910), dated May 5, 1897, was published in the *Journal of the Franklin Institute*, July, 1898, and in the table of tests, covering 82 bars, the maximum increase in strength of treated bars—as compared with untreated companion bars, free from defects—was shown to be 40 per cent., and the maximum increase in deflection was 41 per cent.

Two important practical results have since been developed from the publication of these new principles: One is the great increase in the practice of tumbling castings, wherever practicable, instead of pickling them, as was formerly frequently done, to remove sand burnt into the surface of such castings, because subsequent investigations showed that test-bars and other castings which have been pickled are invariably weaker than companion bars or castings not pickled, the decrease in strength averaging, in the case of test-bars pickled in sulphuric acid and water, about 10 per cent., while castings cleaned in the tumbling-barrel are invariably much stronger than before treatment.

In England, tumbling-barrels of unusually large size have been made and provided with two rates of speed, in order that the full benefit of increase of strength due to mechanical shocks upon castings may be utilized.

In an instructive article (one of a series by Mr. Robert Buchanan, an English writer) on Foundry Management in the New Century, printed in the *Engineering Magazine*, May, 1903, a photographic illustration is shown of such an apparatus;

and testimony of a highly gratifying nature is given of the practical value of these discoveries.

The second important result of the publication of these heretofore unobserved properties of cast-iron, has been the incorporation in some specifications for castings, of a clause stipulating that "the bars shall not be rumbled or otherwise treated, being simply brushed off before testing." The latest instance of the kind may be found in the specifications for gray iron castings,[1] recently formulated by a committee of the American Society for Testing Materials, of which Mr. Walter Wood is chairman.

The concluding sentence of my original paper, The Mobility of Molecules of Cast-Iron,[2] was as follows:

"I have suggested and adopted throughout this paper, the hypothesis of the mobility of the molecules of cast-iron, resulting in an effect similar or analogous to the effect of annealing by heat, because it seems to be warranted by the facts developed by the experiments described; it is of course possible, in our *à priori* ignorance of the laws governing atoms and molecules, that the theory may not be correct, but the tentative propounding of a probable hypothesis, by inciting to a more extended course of experiment, along different related lines of investigation, often leads to establishing or disproving the theory, and thus adds to our stock of positive knowledge."

Professor Ledebur investigated both the facts and the theory advanced by me to explain these facts, and, while admitting the facts, he attributes the astonishing gains in strength to "stretching of the skin of castings," which, he thinks, has come into a state of tension due to the unequal cooling of the test-bar or other casting.

Before the publication of my original paper on the mobility of molecules of cast-iron, I began a long series of investigations from an entirely different point of view, which have, I think, resulted in the discovery of some hitherto unobserved properties of cast-iron (or if observed at all, prior to this announcement, they have not been properly understood), as surprising and unexpected as were the facts contained in the first communication, all tending, moreover, to establish the correctness of the theory then advanced regarding the mobility of molecules of cast-iron.

[1] See page 173.

[2] *Op. cit.*

In accordance with the request of the Secretary of the American Institute of Mining Engineers, received through Mr. Wm. R. Webster under date of November 5, 1903, "that you will take up the subject where you left off in the former discussion and give us the benefit of your present views," I now have the honor to offer for your inspection some test-bars of cast-iron, together with photographs of castings, bars of steel and wrought-iron, and photo-micrographs of sections of such bars, showing remarkable structural changes due to certain treatment described as follows:

The two test-bars of gray cast-iron, submitted herewith, were cast side by side in one mold from one hand-ladle of iron from one runner; the molds were exactly 15 in. long and 1 in. square in section, the ends being formed by iron yokes (machined to 15 in.) inserted in the mold, in order to avoid any possible irregularity in length due to the ordinary method of rapping the patterns before removing them from the mold.

When cleaned, the shrinkage of the bars was measured by inserting a hardened steel wedge between the ends of the bars and a master-yoke, kept for the purpose in a suitable frame, the wedge being graduated so as to show readily thousandths of an inch. (See Fig. 2.) There was a difference of but 0.003 in. in length between the two bars; they measured practically $14\frac{3}{16}$ in. long and 1 in. square in section. One bar remains just as it was cast, and the other has been made to grow gradually in cubic dimensions while in the solid state, until, at the present time, it is $16\frac{1}{2}$ in. long and $1\frac{1}{8}$ in. square in section.

An examination of the milled surface of a portion of the enlarged bar will show a beautiful fine-grained metal with smooth finish, having apparently lost none of its metallic qualities, notwithstanding the extraordinarily large permanent increase in cubic dimensions, exceeding 40 per cent. of the original bulk. The original dimensions of bars Nos. 1 and 2, were $14\frac{3}{16}$ in. by 1 in. by 1 in., equivalent to 14.8125 cu. in., and the new dimensions (bar No. 1) were 16.5 in. by 1.125 in. by 1.125 in., equivalent to 20.8828 cu. in., which showed an increase in volume of 40.98 per cent. This enlarged bar weighed precisely the same as before treatment, but its specific gravity was, of course, considerably less. The actual specific gravity of a piece of a similar test-bar before treatment was found to be 7.13, giving, by calcu-

lation, a weight of 444.75 lb. per cu. ft.; the specific gravity of a piece of the same bar, after having been caused to grow in cubic dimensions about 30 per cent., was found to be 6.01, giving a weight of 375.5 lb. per cu. ft., a difference of 69.25 lb. per cubic foot.

Similar bars of wrought-iron, soft-steel (carbon 0.25 per cent.), tool-steel (carbon 1.25 per cent.), and cast-steel, treated in precisely the same manner as the expanded cast-iron bars, all show a slight contraction, equivalent to about 0.125 in. to the foot.

These curious experiments were commenced in 1896, and early in 1897 Mr. John A. Penton, the editor of *The Foundry*, saw some of the enlarged bars in my laboratory and asked permission to print an editorial note in regard to the matter. The following extract, from the editorial columns of *The Foundry*, April, 1897, entitled "Making Castings Grow," will explain the method adopted at that time for increasing the cubic dimensions of castings without changing their form, weight, or chemical constituents.

"It may be interesting to know that a metallurgical friend of ours, Mr. Outerbridge, of Philadelphia, has been cultivating a very peculiar plant for nearly a year past, which has been steadily growing in the arid soil of a foundry and has not yet reached full maturity. The plant we refer to is a cast-iron test-bar.

"Two bars were cast side by side in one mold from patterns 1 in. square and 15 in. long, between the ends of iron yokes. The bars when cold measured almost the same length, differing only 0.001 in. when placed within the yoke and carefully measured with a graduated steel wedge.

"The shrinkage of one bar was found to be 0.184 in. and the other 0.185 in.

"The shorter bar was then heated by pouring about 10 lb. of molten cast-iron [of the same composition—A. E. O., Jr.] over it, the bar forming the bottom of an open mold.

"On the following day this bar was again measured, and now showed a shrinkage of only 0.15 in. The experiment was repeated, and on a second test the steel wedge showed a space of 0.128 in. between the bar and the yoke. Six times this test was repeated, each time the bar growing longer, and, on the

seventh heating, it would no longer fit between the ends of the yoke, showing that the bar had now grown to the full length of the mold in which the molten iron was originally cast. In other words, the natural shrinkage of the metal had been entirely eliminated by the gradual increase in length due to the successive heatings.

"The bar increased also visibly in thickness, and after the third heating it became crooked, and its position was then reversed each time that it was placed in the mold for heating, in order to obviate this corkscrew tendency.

"After 12 successive heatings and coolings the bar measured a full quarter of an inch longer than its companion, and also lost its tendency to become crooked on heating. After 15 repetitions the bar increased much less than before at each heating, and though it had not finished growing, curiosity to ascertain the effect upon the strength of the metal prompted the experimenter to break both bars upon the testing-machine. The untreated bar broke at a strain of 2,150 lb. (transverse or cross-section stress) and showed a deflection of 0.15 in.; the metal was open-grained, soft foundry-iron, such as is used for casting small pulleys. The companion-bar, which had been subjected to 15 alternate heatings and coolings, as described, broke at 1,250 lb. and showed a deflection of 0.1 in. . . . This curious experiment with the 'growing casting,' which we have described, is not only interesting as a novel investigation, but it casts new light upon the cause of molecular changes in cast-iron when subjected to alternate heating and cooling, and also furnishes valuable information as to the very serious effect upon the strength."

Although complete chemical analyses of both bars were not made, certain chemical tests indicated clearly that the only change in constituents was that a very small amount of combined carbon in this soft metal was converted into graphite by the annealing process. This increase of free carbon (the total carbon remaining the same as before treatment) does not account in any way for the enlargement of the bar. The explanation which I offered at the time and have since corroborated by very many tests, made under varying conditions, is, that the crystals of cast-iron, unlike those of steel or wrought-iron, are capable of inter-molecular movement within a wide range, the full extent of which I have not yet ascertained.

In the more recent experiments, where the cubic expansion of the bar (such as the one exhibited herewith) is more than six times as great as in the bar described in *The Foundry*, of April, 1897, a different method of heating has been employed, whereby a record of the critical temperature necessary to produce the largest results per heat has been carefully kept.

These bars have been heated in a case-hardening furnace provided with a pyrometer, at a uniform temperature of about 1,450° F. for a period of about one hour each day; the gas was then turned off and the furnace allowed to cool down slowly over night, the bars being removed in the morning and measured before re-heating.

In order to prevent scaling, or oxidation of the surface, the bars are enclosed in an iron pipe, the ends being stopped with clay.

At first, a temperature not exceeding 1,200° F. was maintained, and under these conditions it was necessary to heat and cool the bars about a hundred times, in order to obtain an increase in cubic dimensions equal to that shown in the bar submitted herewith. By increasing the temperature to about 1,450° F., an increase in length of about $\frac{1}{16}$ in. per heat is obtained. The bar submitted for your inspection, which shows an increase over its untreated companion-bar of $1\frac{1}{16}$ in. in length and $\frac{1}{8}$ in. in cross-section, has been heated and cooled 27 times.

All grades of cast-iron do not increase in cubic dimensions equally; soft-iron, containing but little combined carbon, increases more rapidly than harder iron containing more combined carbon, while white iron, in which nearly all of the carbon is in the combined form, does not expand sufficiently to overcome the original shrinkage, even after all of the combined carbon has been changed to graphite or rather to that form of free carbon known as "Ledebur's temper carbon" by this annealing-process. Wrought-iron and steel bars subjected to the same repeated heating and cooling, in a closed tube, have all contracted slightly in cubic dimensions. The average contraction of such bars after about 60 heatings in a closed pipe is $\frac{1}{8}$ in. per foot.

In determining the specific gravity of sections about 0.25 in. thick, cut from bars which had been caused to grow in cubic dimensions in the manner stated, a very surprising phenomenon

was observed. After weighing in the air a specimen suspended by a horsehair from one arm of an analytical balance, the piece was immersed in a vessel of distilled water. Instantly bubbles of gas appeared over the entire surface of the metal, rising through the water in a copious stream, the action continuing for more than one hour. On removing the specimen after the bubbling had ceased, drying off the surface-moisture, and re-weighing, a gain in weight, due to absorption of water in the pores of the iron, amounting to 0.870 grams was found. Investigation shows that the bubbles are not "occluded gas" originally entrained, or contained within the metal, but are simply air which enters to fill the void in the spaces between the molecules as the bar cools from the red-hot state.

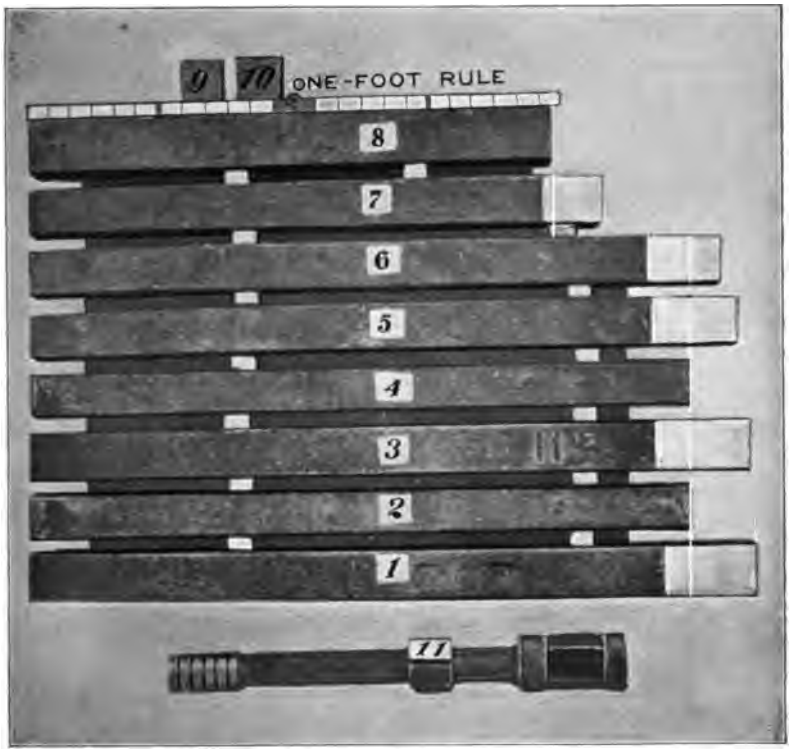
Specimens immersed in gasoline, alcohol, naphtha, and other liquids, all give off bubbles of air in the same manner; and, moreover, if a piece of saturated metal be allowed to remain in a warm room for several hours, the water or other fluid will gradually evaporate and air will again take its place, so that, when the specimen is re-immersed, the bubbling is renewed as actively as before.

As the object was to determine the specific gravity of the mass, not the average specific gravity of the constituents of the metal, the simple plan was adopted of coating the specimen with a thin water-proof varnish before weighing; this protective covering prevented the escape of air, and ingress of water, during the process of weighing, while the specimen was suspended in the vessel of distilled water and the true specific gravity was thus readily approximated.

Notwithstanding the truly astonishing increase in cubic dimensions of these cast-iron bars, none of them have as yet finished growing, nor do they show any signs of disintegration. I am unable, therefore, to define the limits of growth. The experiments are still in progress. (See footnote on page 231.)

The fact that there is a loss of tensile and transverse strength of from 25 to 30 per cent. in some of the tests might seem, at first sight, to put a barrier in the way of a practical application of these new observations; but such is not the case. Already several such applications have been suggested and tried with success. For example, very recently a cast-iron pattern of a "pin-cam lever" was made in two halves for a new machine,

FIG. 1.



Bar No. 1. Total expansion, 27 heatings, is $1\frac{1}{16}$ in. in length and $\frac{1}{8}$ in. in each transverse dimension. Bar No. 2. Untreated companion of bar No. 1. Bar No. 3. Total expansion, 27 heatings, is 1.5 in. in length and $\frac{1}{8}$ in. in each transverse dimension. Bar No. 4. Untreated companion of bar No. 3. Bar No. 5. Total expansion, 50 heatings, $1\frac{3}{8}$ in. in length and $\frac{3}{32}$ in. in each transverse dimension. Bar No. 6. Total expansion, 30 heatings, $\frac{3}{4}$ in. in length and $\frac{1}{16}$ in. in each transverse dimension. Bar No. 7. Original length, 12 in. Total expansion, 49 heatings, 1 in. in length and $\frac{1}{8}$ in. in each transverse dimension. Bar No. 8. (Soft-steel bar). Original length, 12 in. Total contraction, 61 heatings, $\frac{1}{8}$ in. per foot. Bars No. 9 and No. 10. Transverse sections of cast-iron bars before and after heat-treatment. Total expansion of heated piece was $\frac{3}{32}$ in. in each transverse dimension. Bar No. 11. Cast-iron piston having the polished end expanded 0.045 in. by heat-treatment. Bars Nos. 1, 3, 5, 6 and 7 have been chalked at the right hand ends to show the increase in the lateral dimension.

NOTE.—(January 20, 1904). All treated bars represented in this illustration are now larger than when the photograph was taken. No. 1 has been subjected to 12 additional heatings at 1,450° F., and has become increased to 16.625 in. in length and to 1.1406 by 1.1406 in. in transverse section, which is equivalent to a total increase in volume of 46 per cent. The last six heatings made no appreciable increase in the size of the bar.

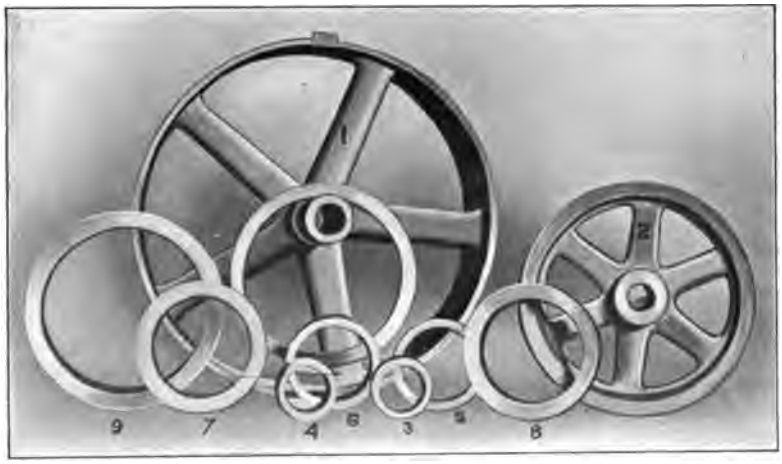
EXPANSION OF CAST-IRON BARS BY REPEATED HEATINGS.

FIG. 2.

The length of the untreated cast-iron bar in the yoke is $14\frac{3}{16}$ in. and the length of its companion bar No. 2, after treatment, is 16.5 in., showing an increase in length of $1\frac{1}{16}$ in. The increase in each transverse dimension is 0.125 in., and the total increase in volume is 40.98 per cent.

EXPANSION OF A CAST-IRON BAR BY REPEATED HEATINGS.

FIG. 3.



No. 1. Cast-iron pulley (turned) $14\frac{3}{4}$ in. in diameter and $\frac{5}{16}$-in. bore. Expansion is $\frac{3}{16}$ in. from four heatings.

No. 2. Blank gear-wheel (turned) $9\frac{3}{16}$ in. in diameter and $1\frac{1}{8}$-in. bore. Expansion $\frac{1}{16}$ in. from five heatings.

No. 3. End of small bushing $2\frac{5}{32}$ in. in outside diameter, $1\frac{7}{64}$ in. in inside diameter. Expansion $\frac{5}{64}$ in. in outside diameter from 20 heatings.

No. 4. Untreated companion piece of No. 3.

No. 5. Piece of bushing $3\frac{9}{16}$ in. in outside diameter and 3 in. in inside diameter. Expansion $\frac{3}{32}$ in. in outside diameter from 15 heats.

No. 6. Untreated companion piece of No. 5.

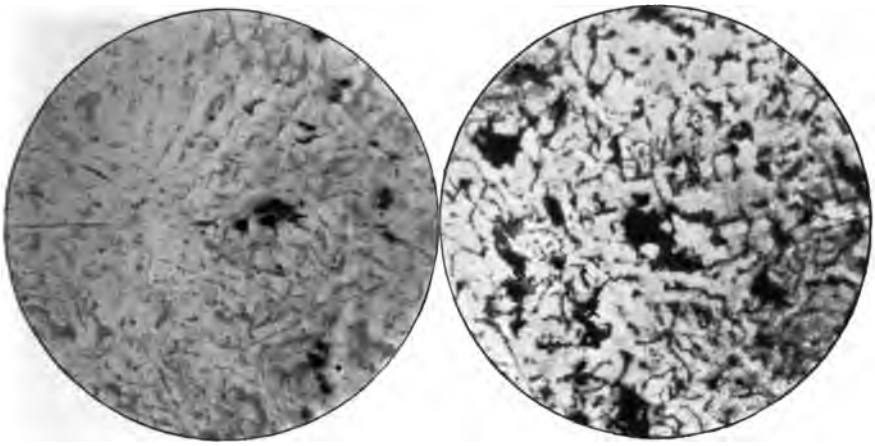
No. 7. Untreated companion piece of No. 8.

No. 8. Piece of bushing 5 in. in outside diameter and 4 in. in inside diameter. Expansion $\frac{3}{32}$ in. in outside diameter from 16 heatings.

No. 9. Piece of bushing $7\frac{7}{16}$ in. in outside diameter and $6\frac{7}{16}$ in. in inside diameter. Expansion $\frac{1}{4}$ in. in outside diameter from 19 heatings.

IRON-CASTINGS EXPANDED BY REPEATED HEATINGS.

FIG. 4.



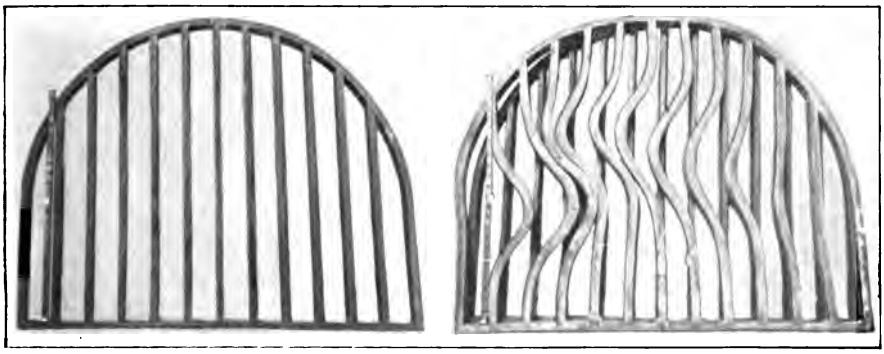
Untreated. Treated.

MAGNIFIED 50 DIAMETERS.

A careful examination of these photomicographs shows a similar arrangement of the crystals in both specimens, the only material difference being the contiguity of the iron particles which have been pushed out in all directions and have not returned to their original positions when cold, thus accounting for the increase in the cubic dimensions of the expanded metal as well as for the ingress of air.

PHOTOMICROGRAPHS OF UNTREATED AND TREATED CAST-IRON.

FIG. 5.



Wooden Pattern. Wooden Pattern and superposed iron-casting made from it.

WOODEN PATTERN AND IRON-CASTING SHOWING THE EXPANSIVE EFFECT OF HEATING.

and after some expensive machine-work had been put upon the castings (they were nearly finished), it was discovered that an error had been made, and that the two half patterns were both a quarter of an inch too short and three thirty-seconds of an inch too narrow. These castings were, therefore, useless, and would have been "scrapped" had it not been for the ingenuity of the operator of the case-hardening furnace in which the experimental test-bars had been heated and expanded, who very successfully elongated and widened these cast-iron patterns, without serious warping or twisting, thereby bringing them in three heatings to the required dimensions.

On page 231 are presented several small photographic illustrations (Figs. 1, 2, 3 and 5), showing castings of various kinds that have been experimentally enlarged by heating and cooling in the manner described. In Fig. 1 a small cast-iron piston is shown (No. 11), which had become worn too small for further use, the maximum allowance for wear being 0.008 in. in diameter. In five heatings, in a closed tube to prevent oxidation, the diameter of the polished end, showing five grooves, was increased 0.045 in., or more than seven times the amount required to restore it to the original size. The illustration shows the piston as it came from the oven before being re-ground to size. It also shows eight test-bars, seven of cast-iron and one of steel, treated in various ways as detailed in the description of the illustration.

Fig. 2 shows an untreated test-bar placed within the iron yoke, the hardened steel wedge in position for measuring the shrinkage. On the table beneath the yoke is shown the companion test-bar, cast of the same dimensions as the other, but expanded $1\frac{1}{16}$ in. in length and $\frac{1}{8}$ in. in cross-section.

Fig. 3 shows a turned pulley, a gear-wheel blank and several bushings and rings of cast-iron, some untreated and some subjected to repeated heatings, as described in detail in the footnote to the illustration.

Fig. 4 shows a photomicrograph of a ground and polished section of an untreated piece of test-bar, and a similarly smoothed and polished section of the same bar after 35 heatings. These photomicrographs, made by Mr. Job, chemist of the Reading Railroad, show very clearly the change in molecular structure of cast-iron, due to alternate heating and cooling.

Fig. 5 shows the expansive effect of heating an iron-casting. The illustration at the left hand shows the wooden pattern from which the original iron-casting was formed, and the one at the right shows the same pattern on which has been superposed the iron-casting that has been expanded by many heatings and coolings during nine months of continuous use. The grating had been laid loosely on the floor of a core-oven, over the flue and nearly 10 ft. above the top of an anthracite coal fire, far removed from flames. The flue was very much smaller in diameter than the casting.

The outside bars (or frame) of the casting resting on the floor and therefore protected from direct radiation of heat, remained comparatively cool and did not tend to expand. The central portion of those bars which were exposed to the greatest heat expanded both in length and thickness, and as the ends were held by the non-expanding frame the bars were forced to assume the curved forms shown in the photograph. The left hand bar of the pattern measured $22\frac{1}{4}$ in. on the longest side, as shown by the two-foot rule placed in proximity thereto.

The corresponding bar of the casting, measured on the same side, showed an increase in length of $2\frac{1}{16}$ in. or 1.11 in. per foot. The bars also increased $\frac{3}{16}$ in. in width in the central portion, the difference in thickness of the center and ends, as well as of the bars composing the outside frame, being plainly shown in the illustration.

Steel does not increase in cubic or linear dimensions by such heat-treatment, and it is proposed to have steel-castings made from this pattern to replace four iron-castings, all of which are distorted in a manner similar to the one shown.

It is a question for future consideration, in view of these facts, whether cast-steel grate-bars will not eventually replace cast-iron ones, although the "scaling" of steel due to oxidation and also to absorption of sulphur from fuel is undoubtedly greater than that of cast-iron under similar conditions.

Tables I., II., and III. give the measurements of various test-bars expanded by heating in closed tubes and in an open fire, and Table IV. shows the data of shrinkages of certain wrought-iron and steel bars.

TABLE I.—*Results of Experimental Heatings of Cast-Iron Test-Bar, Marked "A" No. 1. November 24, 1903.*

	Number of Heats.	Length.	Expansion Since Last Measurement.	Total Expansion.	Rate of Expansion Per Heat.
		Inches.	Inches.	Inches.	Inches.
Before heating		14 15/32			
1st measurement	10	15 1/16	9/32	9/32	0.0284
2d measurement	5	15 5/32	3/32	3/16	0.0187
3d measurement	2	15 9/32	1/8	1/4	0.062
4th measurement	11	15 1/2	7/32	7/16	0.0199
5th measurement	8	15 11/16	3/16	23/32	0.023
6th measurement	4	15 7/8	3/16	1 3/32	0.046
7th measurement	5	15 31/32	3/32	1 3/16	0.0187
8th measurement	1	16 1/16	3/32	1 9/32	0.0937
9th measurement	4	16 3/32	3/32	1 3/8	0.0234

NOTE.—This bar is one of the regular test-bars cast from first of heat of November 24, 1903. It measured 14 15/32 in. long when cast, and yielded a total expansion for 50 heats of 1 3/8 in. per 15 in., or at the rate of 0.0275 in. per heat.

TABLE II.—*Results of Experimental Heatings of Cast-Iron Test-Bar, Marked "C" No. 2. November 24, 1903.*

	Number of Heats.	Length.	Expansion Since Last Measurement.	Total Expansion.	Rate of Expansion Per Heat.
		Inches.	Inches.	Inches.	Inches.
1st measurement	10	12 3/16	3/16	3/16	0.019
2d measurement	5	12 1/4	1/16	1/4	0.0125
3d measurement	2	12 3/8	1/8	3/8	0.062
4th measurement	11	12 9/16	3/16	9/16	0.017
5th measurement	8	12 11/16	1/8	11/16	0.0156
6th measurement	4	12 7/8	1/16	3/4	0.0156
7th measurement	1	12 15/16	1/16	15/16	0.062
8th measurement	8	13	1/16	1	0.0077

NOTE.—This bar is a regular test-bar with chilled ends cut off, and measured exactly 12 in. long before treatment. Middle of the heat of November 24, 1903. Total expansion for 49 heats is 1 in. per foot, or at the rate of 0.0204 in. per heat.

TABLE III.—*Results of Experimental Heatings of Cast-Iron Test-Bar, Marked "E" No. 3. November 24, 1903.*

	Number of Heats.	Length.	Expansion Since Last Measurement.	Total Expansion.	Rate of Expansion Per Heat.
		Inches.	Inches.	Inches.	Inches.
Before heating		14 13/16			
1st measurement	10	14 29/64	11/64	11/64	0.0171
2d measurement	5	15 1/16	5/64	1/4	0.0156
3d measurement	2	15 3/32	1/32	9/32	0.0156
4th measurement	11	15 11/32	1/4	17/32	0.0227
5th measurement	8	15 3/8	1/32	9/16	0.0031
6th measurement	4	15 17/32	3/32	25/32	0.039
7th measurement	5	15 29/32	3/32	13/16	0.0187
8th measurement	1	15 15/16	1/16	15/16	0.125
9th measurement	4	15 15/16	1/16	1	0.0155

NOTE.—This bar is one of the regular test-bars from the last of the heat of November 24, 1903, and measured 14 13/16 in. long when cast. It yielded a total expansion for 50 heats of 1 in. per 15 in., or at the rate of 0.02 in. per heat.

TABLE IV.—*Results of Experimental Heatings of Steel Bars, Soft-Steel, Tool-Steel, Cast-Steel and Iron.*

Material.	Original Length.	No. of Heats.	Contraction.	No. of Additional Heats.	Contraction.	Total Heats.	Total Contraction.	
							Per Foot.	Per Heat.
	Inches.		Inches.		Inches.		Inches.	Inches.
Soft-steel bar, marked No. 2	12	40	$\frac{1}{16}$	23	$\frac{1}{16}$	63	0.125	0.002
Tool-steel bar, marked No. 3	12	28	.....	.....	.....	28	0.023	0.0008
Wrought-iron bar, marked No. 4	12	28	.....	.....	.....	28	0.035	0.0012
Piece of cast-steel test-bar, tested for tensile strength	9	10	.....	.....	.....	10	$\frac{1}{32}$ in. per 9 in.	0.0031

In this report of progress made in the investigations of the molecular structure of cast-iron since my first paper, I have refrained from giving the wearisome details necessary for the investigator to observe carefully, but which merely serve to confuse the reader, who desires solely to comprehend the broad principles.

It is believed that these observations will throw light upon many obscure phenomena which have puzzled founders and others with respect to iron castings that have been subjected to alternate heating and cooling; also that many practical applications of the observations here recorded will suggest themselves to others.

It has long been known that cast-iron grate-bars, and other castings subjected to heat, will warp, twist, buckle and crack, but these effects were attributed in some cases to the burning out of carbon; in others to the oxidation of iron, as well as to the absorption of sulphur from the fuel.

In other cases, as in cast-iron plates used in some glass-annealing ovens, these changes in form and the splitting or cracking of the plates were thought to be merely deformations caused by the temporary expansion of the plates while red hot.

So far as my information and research goes, I believe it has not heretofore been known that cast-iron actually increases in cubic dimensions to an extent not yet finally ascertained, but certainly exceeding 40 per cent. more than the original dimensions, without material change of composition or destruction of its metallic properties. It is also thought that the surprising dif-

ference in behavior of wrought-iron and steel of all grades, when subjected to precisely similar treatment, will prove novel to many engineers and founders; one material expanding greatly, the others contracting slightly under the heat-treatment.

Dr. Charles B. Dudley has recently informed me that, within the past three years, some car-wheel makers have found that the re-annealing of car-wheels enabled them better to withstand, without cracking, the "thermal test" of the Pennsylvania Railroad Co. They have also noticed that such wheels measure as much as $\frac{3}{8}$ in. larger in circumference than before the re-annealing. This is equivalent, on a 36-in. wheel, to $\frac{1}{8}$ in. on 3 ft. of diameter, or far less than the original shrinkage of the metal, and is infinitesimal as compared with the large increase in size of bars here shown, amounting, in one case, to more than 40 per cent. increase in volume. Moreover, a car-wheel is under very great compression-strains due to the chilling of the rim; and it is natural to expect that it would tend to enlarge slightly owing to the more complete relieving of these compression-strains by re-annealing. The wheels would, no doubt, continue to increase in size if re-annealed many times, as is the case with expanded bars.

It has also been suggested that the conversion of combined carbon into the graphitic form may have something to do with the enlargement of the castings. One answer to this—as already given—is, that white-iron bars, in which practically all of the carbon is combined, do not expand, even after the breaking-up of the whole of the combined carbon and its change into graphite, or, more correctly speaking, Ledebur's "temper" carbon (as in some modern malleableizing methods), sufficiently to overcome the original shrinkage of the metal. Furthermore, soft-iron bars, having more than 2 per cent. of silicon and very little combined carbon, expand much more under similar heat-treatment than harder gray iron containing much less silicon and much more combined carbon.

Dr. Dudley has also recently suggested that "absorption of oxygen" may possibly have some influence on these changes. In experiments which I made many years ago of casting iron in a mold faced with "carbonized fabrics," (such as an embroidered lace veil in which the carbon threads were as fine as cob-webs), I found that there was little or no tendency for the

carbon to be consumed; showing that the oxygen of the air was driven off by the heat through the mold. It would seem, therefore, that in the first experiments in heating and expanding cast-iron bars by pouring molten iron over them, there could have been no absortion of oxygen. In several of the more recent tests I have filled the iron tubes containing the bars to be expanded with charcoal, rammed tightly into the interstices between the bars in order to prevent oxidation, and found no appreciable difference in expansion from bars heated in a closed tube without charcoal. I have, however, contended for many years[1] that iron melted in a cupola absorbs oxygen from the air-blast; and I have attributed the marked difference in quality between high chilling-iron melted in a cupola and the same iron melted in a reverberatory furnace largely to this cause. In my discussion on pig-iron,[2] the following sentence may be found: "Finally, I may say that I am convinced that the presence of iron oxide dissolved in the molten metal is a factor which is commonly overlooked, and is to be feared, especially in strong iron mixtures. The best deoxidizer with which I am familiar is ferromanganese, a metal containing about 80 per cent. manganese, which may be added either in the ladle or in the cupola in the proportion of about 1 lb. to 600 lb. iron. The effect is to increase the strength and ductility, to decrease the chill, and to darken the gray color of the fracture in high-chilling iron mixtures. This discovery was first brought to the attention of metallurgists in an address given at the Franklin Institute in February, 1888, on "Pig-Iron, Including the Relation between Its Physical Properties and Its Chemical Constituents." Since that time the beneficial effects of ferromanganese as a deoxidizer and desulphurizer in certain iron mixtures have become generally known, and its extensive use, especially in car-wheel foundries, has followed thereupon."

All of the investigations so far made in expanding cast-iron by heat, whether by pouring molten iron over cold bars, heating in a closed tube with or without charcoal, heating in a fire without any protection, or by direct radiation of heat over a fire far removed from the flames (as in the protective grating shown in Fig. 5), produce practically the same results, and tend

[1] See lecture on Pig-Iron, *Journal of the Franklin Institute*, March, 1888.
[2] *Idem*, November, 1900.

to convince me that the astonishing change in volume is a molecular, not a chemical, one; thus substantiating my original theory of the molecular mobility of cast-iron.

The peculiar property of cast-iron of increasing in bulk under the influence of heat is inherent in the metal, and has existed for all time. It must, therefore, have been noticed to some extent long ago; but, so far as I am aware, no careful study has ever been made of the phenomenon, if it was observed, and no previous knowledge has, I believe, existed of the fact that a bar of iron can be increased in bulk more than 40 per cent. over its original volume while in the solid condition and still retaining its metallic properties. The decrease in strength appears to bear a certain relation to the degree of expansion of the test-bars.

In conclusion, I wish to repeat what I said in my first communication in 1896, viz.: "My paper, therefore, is not presented as an exhaustive, but as a tentative treatment of this interesting and, I believe, novel line of research, which is not incapable, even in its present stage, of some practical applications." I desire, furthermore, to repeat that I consider that these investigations afford corroborative proof of the correctness of the theory concerning the mobility of molecules of cast-iron, noted in my paper of February, 1896.[3] I also wish to acknowledge my indebtedness to Mr. William H. Maw, Editor of *Engineering* (London), for some valuable suggestions communicated during the past six months.

POSTSCRIPT (communication to the Secretary)*:—I have already remarked in my paper under this title, read at the Atlantic City meeting, that the specific gravity of a specimen of expanded cast-iron was found to be 6.01 as compared with 7.13 for the untreated metal, making a difference of 69.25 lb. per cu. ft. Similar tests more recently made show a still greater decrease in density, that is, a piece of a test-bar which had the same specific gravity before treatment (7.13) was reduced to 5.49 sp. gr. after having been heated 25 times in a closed tube; the weight of a cubic foot of this metal is, therefore, 343.2 lb., making a difference of 101.55 lb. per cu. ft. between the treated and untreated metal.

[3] *Trans.*, xxvi., 176.

* Received March 18, 1904.

Quite recently Messrs. Williams, Brown & Earle of Philadelphia have made for me some interesting "radiographs" with radium, which show that emanations from this substance will penetrate cast-iron readily. Two pieces of a cast-iron test-bar were cut to $\frac{1}{16}$ in. thickness, one before and the other after expansion. These were laid upon a thin strip of lead (a metal which is almost impermeable to the radium rays) cut to a sort of dumb-bell shape. A tube containing 100 mg. of radium of 10,000 activity, in Curié's scale, was placed centrally 4 in. above the pieces of cast-iron. After an exposure of 38 hours, the negative, which was placed beneath these objects, was developed and a print made therefrom. The rays passed through both pieces of cast-iron and were intercepted by the lead strip, making a distinct image of the lead strip underneath the cast-iron. The fact that such dense metal as iron and steel are permeable to emanations from radium may prove of practical value in detecting flaws.

In the course of these studies several interesting observations of minor importance have been recorded, some of which show very clearly the remarkable difference in the effect of repeated heating and cooling of cast-iron, wrought-iron and steel. Others show the difference in effect of repeated heating and cooling of soft cast-iron containing over 2 per cent. of silicon with very little combined carbon, and of harder iron low in silicon and comparatively rich in combined carbon. As an illustration of the former, I may refer to two cases. First: One end of a steel pipe, used to inclose the cast-iron test-bars, was closed by pouring molten pig-iron into it, in order to make a plug about 1 in. thick; the other end was stopped with clay. The pipe containing the test-bars was subjected to repeated heating and cooling, with the unlooked-for result that the cast-iron plug expanded and split open the end of the steel pipe. Second: A piece of wrought-iron pipe 17 in. long was used to enclose some of the test-bars shown in Fig. 1, which were $14\frac{13}{16}$ in. long when cast; the ends of the pipe were stopped with clay. After several heatings the cast-iron bars increased in length to such an extent that the pipe (which contracted slightly in length) was found to be too short to permit of closing the ends with clay, and it was necessary to obtain a longer piece of pipe to continue the experiment in expanding the cast-iron bars.

As an illustration of the difference in effect of repeated heating of soft iron, and of "chilling iron" containing combined carbon, the following case may be cited. A test-bar of strong iron, low in silicon, was cast in a sand mold against the ends of an iron yoke placed in the mold, the ends of the bar being chilled white to a depth of about 0.25 in. When heated, the white iron of the ends was annealed to gray, as in some of the methods employed in making malleable cast-iron, but—and here comes the interesting feature—the gray iron thus made at the ends of the bar did not continue to increase in cubic dimensions when reheated and cooled, as did the normal soft gray iron. The result of the treatment was that the ends of this bar formed themselves in the shape of a cup; the ends were then cut off, leaving a bar 12 in. long. The heat-treatment was continued, and the bar still increased in cubic dimensions, but showed no more tendency to cup. In another case a different effect was produced; the ends did not cup, but the annealed mottled iron of the ends increased in dimensions less than the normal gray iron in the body of the bar, so that the section at the ends was appreciably smaller than that in the main portion. Bars Nos. 5 and 6 in Fig. 1 show this effect.

During the period of these investigations, it happened that one of a pair of steel rocking-bars, about 16 in. long, made for part of a type-casting machine, was found to be a few thousandths of an inch too long to fit into the frame; it had a number of lugs spaced, like its companion-bar, at regular intervals, the whole being cut out of a solid plain steel-bar, on a milling-machine. It could not be shortened by simply milling off the ends, because this would have thrown the lugs out of true relation to the other parts of the machine. The piece was therefore condemned to be "scrapped," but the operator of the case-hardening furnace, having observed that the wrought-iron and steel bars which he had treated for me decreased slightly in length, carefully heated the condemned piece a few times and reduced it to the exact length required; the contraction was so uniform that the lugs were not measurably out of their true relation; the piece was inserted in its frame and found to be perfect in operation. The two rocking-bars operate in unison, being connected by long steel rods sliding perpendicularly in fixed collars without any lateral motion. But for this treatment

it would have been necessary to make a new rocking-bar, and probably two of them, at a considerable cost.

Shortly after my first paper for this meeting had been put into type, the disastrous fire occurred in Baltimore which necessitated a change of place for the meeting of the Institute. It also incidentally furnished striking evidence of the vital importance of a clear understanding of the laws governing the widely different behavior of cast-iron and steel when subjected to prolonged heating. While we deplore the great catastrophe, we may learn the lesson which it teaches, and appreciate the fact that here is presented a timely and practical illustration, on a large scale, of the principles based upon experiments on a small scale which I have discussed in my paper.

Several weeks ago I received a letter, dated Feb. 9, 1904, from an officer of the Land Title and Trust Co. of Philadelphia, which owns and occupies two of the largest and tallest steel structures in that city, to the following effect: "The Chief Engineer of the Land Title Building, who went to Baltimore to inspect the effect of the conflagration upon the Continental Trust Company's 16-story building, tells me that while the entire inflammable contents of that building were 'roasted' and consumed, the steel frame-work is apparently plumb and intact. He also says that the cast-iron frame-work of the doors, which was set in granite, has expanded to a considerable extent and has not again contracted. He says this permanent expansion is very noticeable."

Is it not reasonable to infer that if the frame-work of the doors and windows of this modern so-called fire-proof "sky-scraper" building had been of light steel-construction, and the doors sheathed with thin sheets of steel, or made of non-combustible material, the valuable contents of the building might have been saved from complete destruction by fire from without? At all events, I think it may be assumed that if the laws governing cast-iron and steel when exposed to prolonged heating had been thoroughly understood by architects and constructing engineers, the frame-work of the doors of these modern steel-buildings would not have been made of so unsuitable a material as cast-iron, and I anticipate that, in future constructions of this character, these errors will be avoided in view of our present knowledge and experience.

If steel behaved like cast-iron under such conditions as prevailed during the Baltimore fire, I am of the opinion that the steel-skeleton structures instead of standing to-day "apparently plumb and intact" would have toppled over during the conflagration, owing to unequal growth in dimensions due to unequal heating in parts.

Stock-Distribution and Its Relation to the Life of a Blast-Furnace Lining.

BY DAVID BAKER, SYDNEY, NOVA SCOTIA, CANADA.

(Atlantic City Meeting, February, 1904.)

WHEN the skip-hoist was first tried as a means of filling the blast-furnace it made a great many enemies and very few friends among furnace managers. This state of affairs continued until the Duquesne furnaces were built, which demonstrated not only the possibilities of a furnace-stack in the way of production, but also the ability of the skip-hoist to charge the materials necessary to produce the large tonnage obtained. Every furnace manager, who has operated stacks fitted with the vertical hoist and the hand-barrow system of filling, remembers that the momentous question about the furnace, particularly during a good run, was: "Are you keeping her full?" and many a good run has been spoiled by the furnace "getting away from the men."

With the new birth of the skip-hoist this trouble was eliminated; but alas! Other and more important difficulties arose. The distribution of the stock, the importance of which is never learned until many different ores and fuels have been tried, demanded closer attention. With all the defects of the hand-barrow system, the distribution—when the top fillers were conscientious in carrying out the scheme of filling arranged after thoughtful consideration by the manager—was above criticism. It is said that the modern skip-hoist, equipped with an elliptical hopper and receiving-cylinder, and discharging over a small bell into the main hopper, will give just as good results as those obtained with hand-barrow charging, but I cannot agree with this statement. I grant, that under very favorable conditions, the mechanical furnace-top will do apparently as well as the

hand-system of charging, but under unfavorable conditions it fails. It is the unfavorable case that is all-important, for that frequently means the success or failure of the business concerned.

I am now speaking from my own experience. Possibly there is in the market to-day a mechanical top-charger that will give as good distribution as that obtained by the hand-barrow system, but the low fuel-consumption obtained by the furnaces equipped with the old system has never, to my knowledge, been equalled by those with the new. Can any one point to such a record by a new plant as that reached by Furnace No. 6 of the Illinois Steel Company, which showed for a year a fuel-consumption of 1,581 lb. of coke per ton of pig-iron produced?

I believe the reason for this lies in the great abuse to which coke is subjected in these new plants. I refer to the large number of times the coke is required to drop from one receptacle to another, grinding much of it into dust and breaking the balance into pieces much smaller than those of the original coke freshly drawn from the oven. It is quite common at such plants, for the coke to be forked into hopper-cars at the ovens, dropped from these into a deep steel bin at the furnace, drawn into the larry-car, dumped into the skip, thence into the top receiving-hopper, from which it drops on to the main bell, and is finally discharged into the furnace. Other modern plants avoid one dump by sending the coke through chutes direct from bins to skip-cars. At first sight, this seems to avoid one handling and to save a great deal of breakage; but usually these bins are so much deeper than those in connection with the larry that the breakage is nearly as great.

By this mechanical system the coke is dropped from six to seven times as compared with three or four times by the hand-barrow system, and one of these (from the barrow into the furnace-hopper) can hardly be called a drop, the breakage resulting therefrom being so slight.

Many of my readers have seen what, ten years ago, might have been called a modern plant, equipped with hand-barrows which take the coke direct from box-cars into barrows. In this system the coke receives practically only one drop that is likely to cause any breakage, *i.e.*, when it is discharged from the bell into the furnace.

This careful handling of the coke is of the utmost importance,

and when the coke is of a soft or friable nature, it may mean the difference between the success and the failure of the blast-furnace operation.

The baleful results of the breakage by the repeated handlings of coke in the modern furnace-plant equipped with the usual mechanical charger, as well as the necessity of obtaining good distribution of the stock, were demonstrated very clearly at the plant of the Dominion Iron & Steel Company, at Sydney, Nova Scotia.

The coke, made in the Otto-Hoffman by-product ovens about 1.5 miles from the furnace plant, is delivered to the furnaces in hopper-cars each of 20 tons capacity, and is dumped from them into large coke-bins at the foot of the skip-hoist; thence it is drawn direct into the skip-bucket.

The hoist is a double-skip arrangement, and the top charging-device, shown in Fig. 1, consists of a circular hopper, having the discharge controlled by a small bell.

The coke used at this plant was made from the Dominion Coal Company's coal, and was very friable and quite soft.

The first furnace was blown-in very successfully, Feb. 4, 1901, and everything worked smoothly for a few weeks; but, at the end of this period, the furnace began to hang and slip, and burn the tuyeres, sometimes cutting at a slip every tuyere in the furnace. The tuyeres failed in nearly every case on the lower side of the nose, showing that they must have been flooded by cinder and iron when the slip occurred. The destruction was so rapid that, in some cases, the tuyeres exploded, blowing down the "blow pipes" and the "pen stocks," causing a panic among the furnace men and not adding to the comfort of the furnace manager.

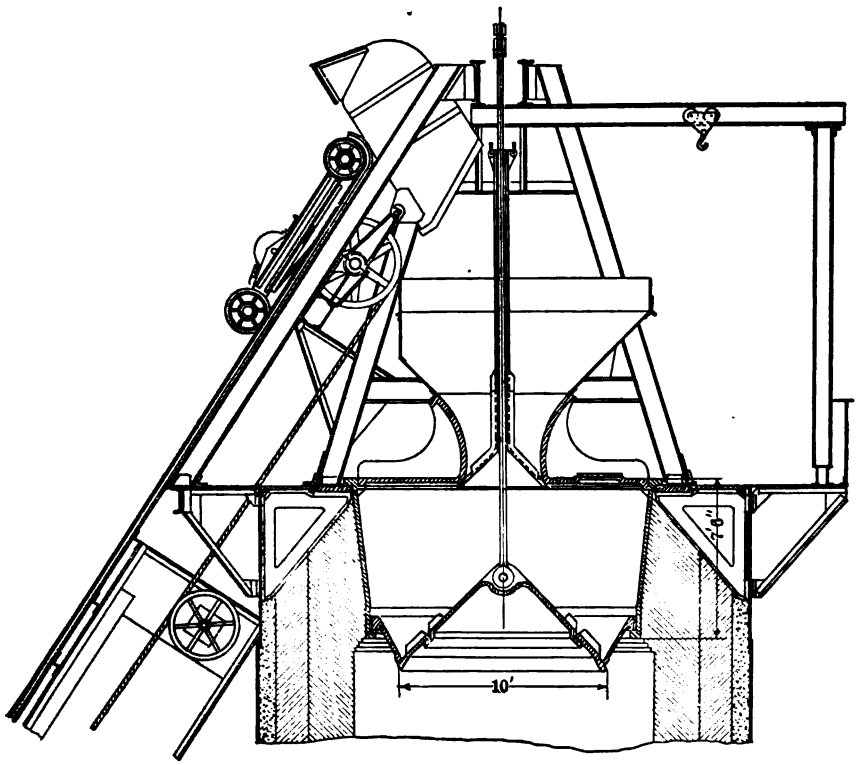
An analysis of the flue-dirt gave the following results, which show that most of it was coke:—Loss on ignition, 52.87; SiO_2, 8.60; Fe_2O_3, 16.35; Al_2O_3, 6.56; CaO, 5.51; and MgO, 3.46 per cent.

The coke in the bins was very dirty and badly broken, and it was apparent that the coke in the hearth of the furnace was so finely divided that the slag found great difficulty in settling through the mass, particularly during a slip; hence, the flooding of the tuyeres with cinder and iron, and their consequent destruction.

In fact, it was only after long blowing at the cinder-notch, and repeated checkings of the blast that the tuyeres could be dried sufficiently to take off the blast without filling the blowpipes.

The first effort made to reduce the tuyere-destruction was the increase of the feed-water pressure by changing from the regular tank-service of salt water at 20 lb. to the fresh-water main at 40 lb. per sq. in. This made only a slight improvement.

FIG. 1.



TOP OF FURNACE NO. 1. FIRST CAMPAIGN.

Next was the steady increase of the number of engine-revolutions, until an average of 36,000 cu. ft. of air per minute, measured by piston-displacement, was forced through the tuyeres. After this change the slipping was less frequent, the output greater and, at the same time, no more tuyeres were lost.

Finally, silica rock was introduced in the charge, which had a marked effect, making the slag very fluid, reducing the burn-

ing-out of the tuyeres materially and increasing the output greatly. The best results were obtained by the addition of 200 lb. of quartz-gravel for each ton of ore charged, changing the slag-analysis from 31.54 to 34.32 per cent. of SiO_2, and from 15.2 to 12.76 per cent. of Al_2O_3. At that time, dolomite was used as a flux.

The ore obtained from Wabana, Belle Isle, Conception Bay, Newfoundland, is a uniform, dense, red hematite, which has been split up *in situ* by cleavage-planes into lumps of rhomboid shape, having a similar appearance to broken red-brick.

The ore is practically free from fines, and is of the following average analysis:—Fe, 51.43; SiO_2, 14.03; Al_2O_3, 4.57; MgO, 0.52; CaO, 2.23; Mn, 0.15; P, 0.805; S, 0.038; and TiO_2, 0.45 per cent.

The high percentage of alumina in the ore, which seems to render the slag viscous, is only corrected by the addition of silica, the best results being obtained when the slag does not exceed 13 per cent. in alumina. Probably the titanic acid also adds considerably to the infusibility.

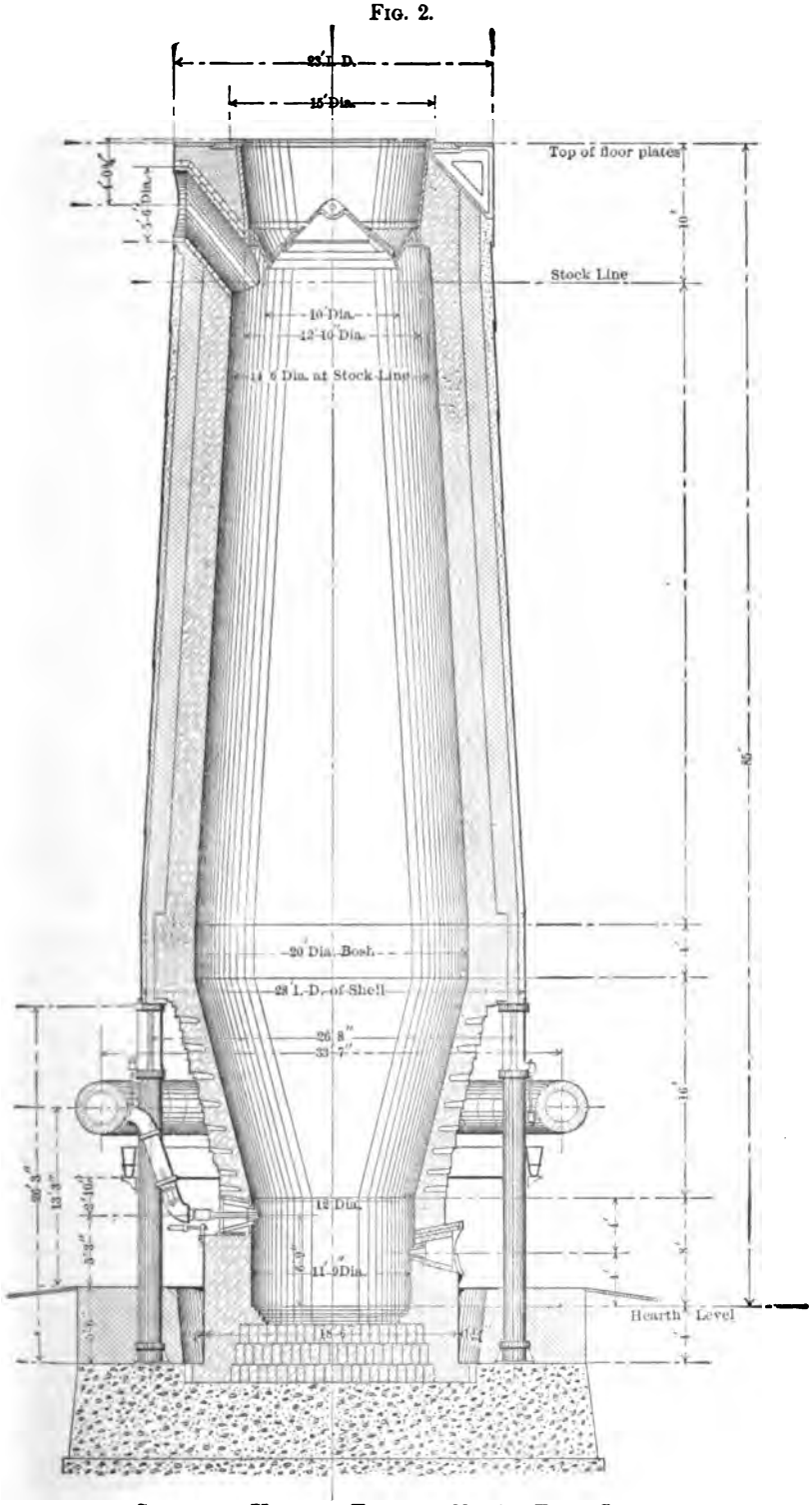
Notwithstanding all the changes previously described, which lessened the destruction of the tuyeres, there was still room for much improvement. Screens were placed in the chutes from the coke-bins to the skip-buckets, which eliminated 4 per cent. of fine coke-braize.

These changes had hardly been accomplished when the shell of the first furnace began to show an unusual temperature about 20 ft. from the top of the bosh and opposite to the skip-way. The furnace had been in blast only six months, and the bricks, made by Harbison & Walker, of Pittsburg, Pa., were held responsible for this rapid failing of the lining. It was said by those present when these linings arrived that the bricks had not been properly housed, and had been subjected to freezing after being wet. Fig. 2 shows the first lining, in which the bricks used in backing up the in-wall were of second quality Scotch make. The "burned" lines of this furnace, Fig. 3, shows that the failure of the in-wall was confined to the side opposite the skip-way.

The second furnace, No. 2, started five months after the blow-in, and developed the same weakness in exactly the same place.

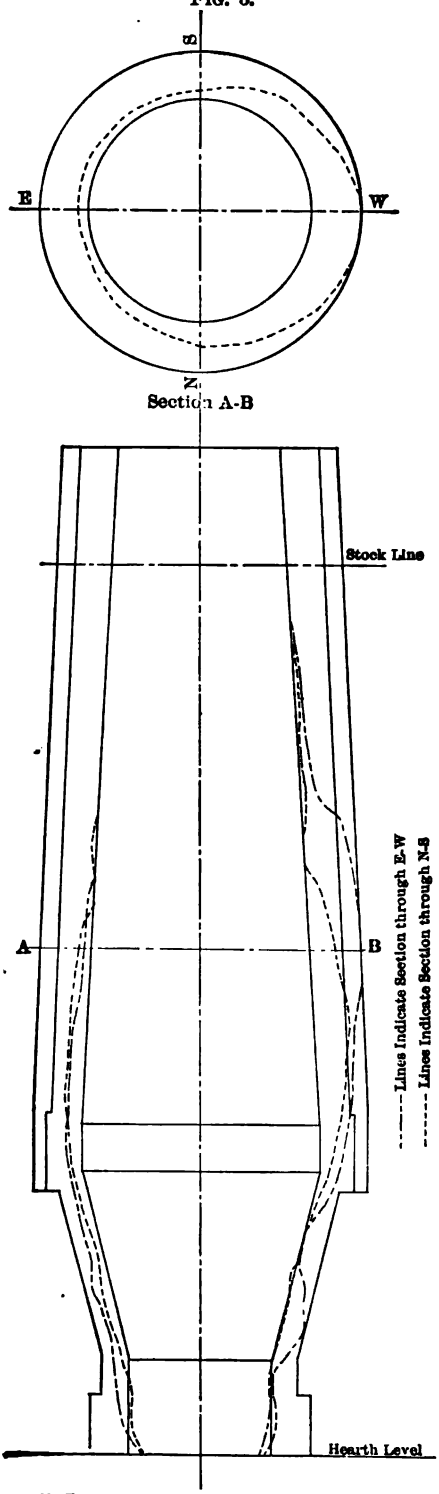
From the first I doubted that the bricks were wholly respon-

FIG. 2.



SECTIONAL VIEW OF FURNACE NO. 1. FIRST LINING.

FIG. 3.



"BURNED" LINES OF FURNACE NO. 1. FIRST LINING.

sible for these failures; and when the third furnace, after a most favorable start, with all the conditions normal, developed a hot spot within three weeks after blow-in, and cut through nearly 5 ft. of fire-brick in exactly the same place, I was convinced that the true cause of the trouble should be sought in another quarter. The condition of the coke charged and the distribution of the charge had been carefully studied for some time, and it became evident that they had some connection with the irregular wear of the furnace in-walls.

When the first lining failed, a model furnace was prepared with a complete top-charging arrangement, the shell being of glass strips fastened to narrow wooden strips, so that the descent of the charges could be observed. The material was drawn from the bottom so as to lower the stock in the same way as the blast-furnace charge is moved by the combustion of the coke in the hearth. At the same time a careful study was made of the actual filling in operation at the furnaces.

The result of this observation showed that the furnace-hopper was filled in the manner illustrated in Fig. 4; the finer portion of the charge remained under the dumping-point of the skip, while the coarser portion of each material rolled to the other and the same side of the furnace where the lining had failed in every case.

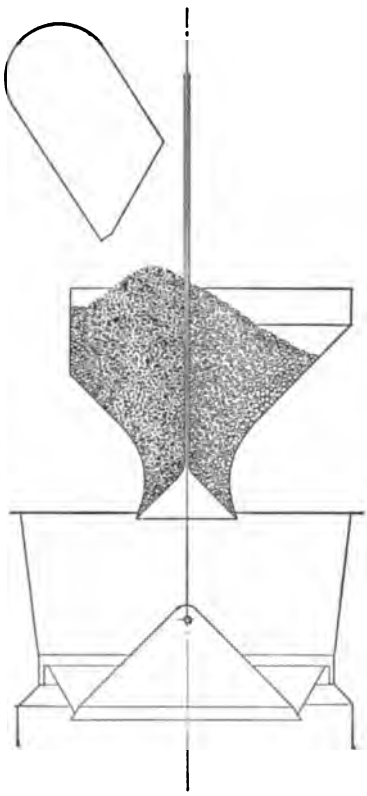
The furnace, therefore, had on one side the best coke and all lump-ore and flux, and on the other side, the coke-dirt and fine coke, together with the finer portion of the ore and flux, which latter material had been crushed at the quarry to pass through a 4-in. ring. As a result of this condition the furnace worked faster and hotter on the side having the most lumps; in fact, it was "channeling" in that direction. Had the charge been of regular Lake Superior ore, with about 10 per cent. of lump-material, and Connellsville coke, possibly no such separation would have been noticed, for there would have been enough fine ore to make the charge practically of the same penetrability to the upward-moving gases, and cause, at the same time, much less breakage of fuel, so that in efficiency the whole charge would have been more nearly uniform.

In order to correct these defects of distribution, a top for Furnace No. 2 was designed, as shown in Fig. 5.

In this new furnace-top an elliptical hopper replaced the old

circular one, the idea being to prevent the sizing of the material by discharging this elliptical hopper into a cylinder closed by a bell. The head-frames of the top were so low that the cylinder could not be made as long as was desired, and it did not have quite the capacity of two full skips. Four months after the "blow-in," this furnace developed a hot spot on the

FIG. 4.



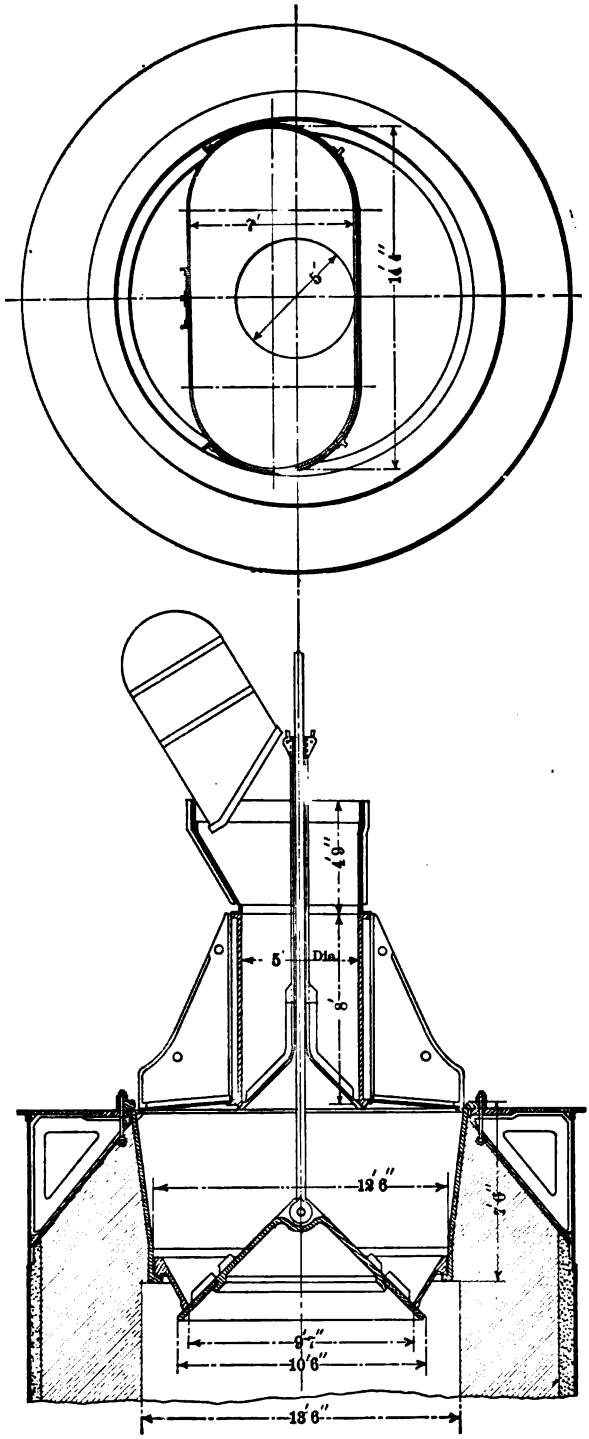
THE DISTRIBUTION OF ORE IN THE STOCK-CHUTE OF FURNACE NO. 1. DURING THE FIRST CAMPAIGN.

side opposite the one previously affected, indicating that the sizing had been simply reversed, but not corrected.

The second attempt to correct the distribution without entirely removing the head-frames and changing the angle of the skip-way is shown in Fig. 6.

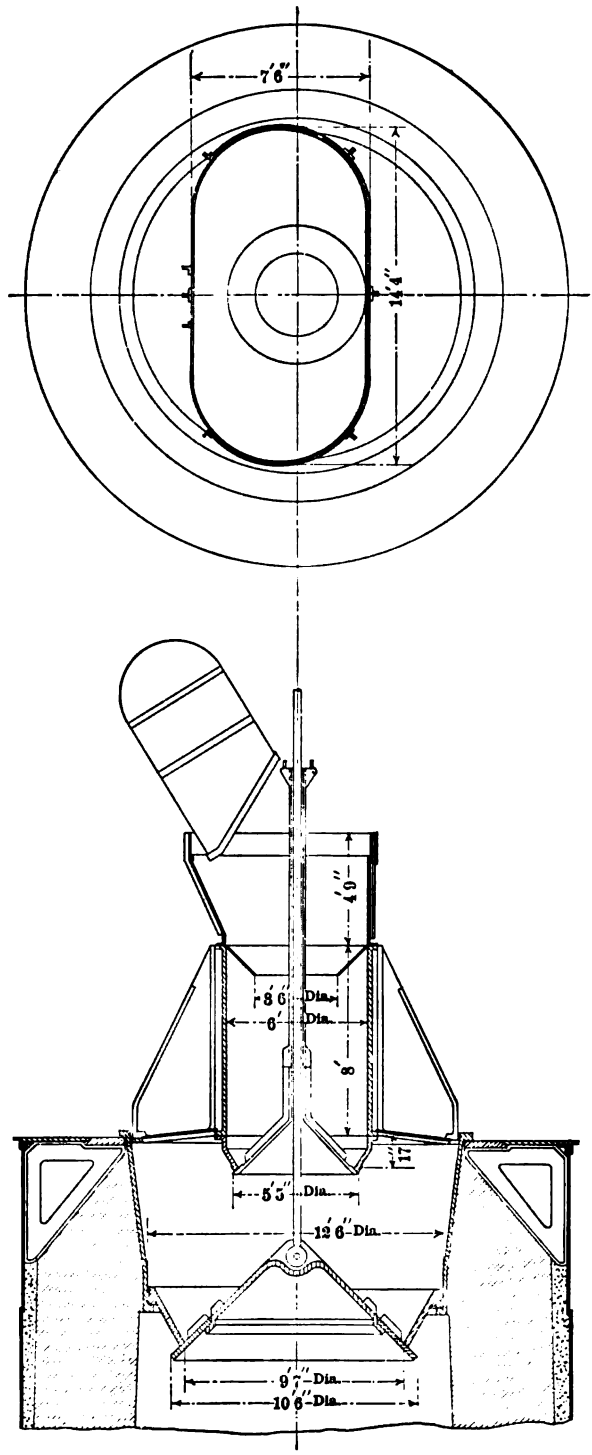
In order to make room in the cylinder for two full skip-loads of coke, the diameter was increased to 6 ft., and the lower part

FIG. 5.



TOP OF FURNACE NO. 2. SECOND CAMPAIGN.

FIG. 6.



TOP OF FURNACE NO. 3. SECOND CAMPAIGN.

of the elliptical hopper was extended into the cylinder, thus narrowing the opening to 3.5 ft., so as to throw all of the material to the center of this cylinder, and in the sizing, which must inevitably take place, to leave the fines in the center and the lumps next to the shell of the cylinder. This furnace has now been in blast 18 months and has not shown any unusual heating at the shell.

The breakage of the coke continues and cannot be remedied with such a filling-system, but enough has been done to show that with washed coal well coked, a fairly uniform furnace-practice can be maintained with only the normal destruction of tuyeres. Unquestionably, the excessive breakage of the fuel very materially lowers its practical calorific efficiency in the blast-furnace. This efficiency is in inverse proportion to the breakage, for, other conditions being equal, the amount of carbon absorbed by the hot CO_2 of the gases is in proportion to the surface exposed. This means that, other things being equal, the fuel-consumption in a blast-furnace per ton of pig-iron produced is directly proportional to the breakage the coke receives between the oven and the hearth of the furnace. Therefore, in laying out new works it is of the utmost importance to study the blast-furnaces and coke-ovens together from this standpoint, in order to put the coke into the furnaces with the minimum amount of handling. This amount may be reduced with the right design to only two drops.

In recapitulation, this experience demonstrates that greater care is needed to distribute well an ore that is all lumps, particularly when smelting with soft coke that is subjected to the abuse of the repeated handlings incident to the modern filling- and charging-device. Whether the labor saved by such filling-arrangements compensates for the loss due to impaired efficiency of the fuel resulting from such abuse, is a question left for others to solve.

The Plotting of Sizing-Tests.

BY W. SPENCER HUTCHINSON, BOSTON, MASS.

(Atlantic City Meeting, February, 1904.)

THE experiments described in this paper were undertaken primarily for the purpose of measuring the quality of work done in screening and sorting in American concentrating-mills for Prof. Richards' work on *Ore-Dressing*. For this purpose a series of samples of screened and sorted products was obtained from four different mills, and a plan was devised for sizing these products, and for tabular and graphic representation of the results. The results, together with discussions as to their significance, so far as they serve to interpret the quality of mill-work, have already been published in the above-mentioned work, and it is not the purpose of this paper to duplicate the work, except so far as is necessary to explain and illustrate methods and appliances.

The sizing of coarse products was done by sifting on screens of sheet-metal with round punched holes. These holes ranged in size from 64 mm. to 0.5 mm. in diameter. Below the smaller limit, punched metal, suitable for this work, was not obtainable, and the sizing was carried on by sifting on woven wire-sieves. The finest woven wire-cloth obtainable gave a hole 0.073 mm. square, and the finest silk bolting-cloth gave a pentagonal hole with an average diameter of 0.069 mm. Even these very fine screens served no useful purpose in sifting certain of the very fine classifier-products, of which all, or nearly all, passed through the finest screen. A new adaptation of a method of settling and decantation with water in beakers was therefore worked out, and by this means the sizing was carried down to grains of a diameter as small as 0.012 mm.

Description of Apparatus and Manipulation.—Sieves of sheet-metal with round punched holes were used as far as practicable, chiefly because, as already explained, the primary object of the experiments was to test mill-work, and the mills had all used

sheet-metal with round punched holes for coarse sizing and, further, because the diameter of a round hole is its only limiting dimension, whereas a square hole may limit the passage of a grain of ore, according as the grain approaches, to take advantage of the diagonal dimension or the side dimension. The Rittinger sieve-scale was adopted as a standard for these sizing-tests, and screens were secured which would match the sizes of the Rittinger scale just as closely as possible. The sizes of the Rittinger scale are in geometrical progression. Diameters of successive sizes have a constant ratio of 1.414, while the areas of holes of successive sizes have a constant ratio of 2.

The screens coarser than 2 mm. were measured with a scale, using a magnifying glass. Screens with holes 2 mm. in diameter and smaller, were measured under a microscope, using a micrometer. From 10 to 20 holes were measured in each screen, and the average of these measures was taken to represent the size of hole for that screen. Sizes of square holes were determined by the mean of the measurements of the two sides. Some of the holes in the woven wire, instead of being truly square, were trapezoidal. In this case the long and short parallel sides were measured, and their mean length taken for one dimension. The sizing-scale used is shown in Table I. The first column shows diameters in millimeters of the Rittinger sieve-scale, the third column shows the actual diameters of the holes in screens obtained for the work in these experiments.

The samples were sifted dry, and the shaking and jarring on the sieves were carried on for such a time that the resulting weight of any size, expressed in per cent., should be significant in the first place of decimals. By trial it was found that on the punched screens, a 4-minute shaking was usually sufficient, while often on the woven wire-screens two or three times as long was needed; the time depending on the size of the sample and the area of the sieve.. In sizing the finer products, it was found that samples weighing 200 grams for 1 sq. ft. of screening-surface could be handled in a reasonable time, requiring somewhat less than 10 minutes vigorous shaking and jarring.

In selecting a method of sizing for the material finer than 0.069 mm., we considered several methods. The use of the assay *spitzlutte* was not practicable, because of the very small

water-currents necessary, the tediousness of the work, and the difficulty of separating the products, for weighing, from the relatively large bulk of water introduced. Certain experiments made by Prof. W. O. Crosby[1] in the sizing of samples of clay were considered. He introduced a weighed portion of material in the top of a 0.5-in. glass tube, 5 ft. long, filled with water. The depth of the sediment was then measured at the end of different periods up to one hour, and these depths were assumed

TABLE I.—*Sizing-Scale.*

Sizes Sought for. True Rittinger Scale.		Sizes Actually Obtained and Used.		
Diam. of Hole.	Log. Diam. Multiplied by 6.64.	Diam. of Hole.	Log. Diam. Multiplied by 6.64.	Means of Obtaining the Sizes.
Mm.		Mm.		
64.0	12	63.5	11.98	Punched sheet-steel; round hole.
45.2	11	44.3	10.94	
32.0	10	31.9	9.99	
22.6	9	22.3	8.96	
16.0	8	16.0	8.00	
11.3	7	11.2	6.97	
8.00	6	8.02	6.01	
5.66	5	5.61	4.98	
4.00	4	3.94	3.96	
2.83	3	2.69	2.85	
2.00	2	1.89	1.84	
1.41	1	1.49	+1.15	
1.00	0	0.945	—0.16	
0.707	—1	0.667	—1.17	
0.500	—2	0.493	—2.03	
0.353	—3	0.371	—2.86	
0.250	—4	0.270	—3.78	Woven-brass wire-cloth; square hole.
0.177	—5	0.158	—5.32	
0.125	—6	0.119	—6.14	
0.088	—7	0.078	—7.55	
0.0625	—8	0.069	—7.7	Silk bolting-cloth; pentagonal hole.
0.044	—9	0.047	—8.8	Smallest quartz-grain settled in water in 15 seconds.
0.0312	—10	0.034	—9.8	Smallest quartz-grain settled in water in 30 seconds.
0.022	—11	0.025	—10.6	Smallest quartz-grain settled in water in 60 seconds.
0.0156	—12	0.019	—11.4	Smallest quartz-grain settled in water in 120 seconds.
0.011	—13	0.012	—12.8	Smallest quartz-grain settled in water in 300 seconds.

to represent proportional weights. The sediment was then excavated from the tube to the points where measurements were taken, samples removed and the grains measured with a microscope-micrometer. This method was discarded chiefly because the volumetric determination of quantities would be inapplicable to mixtures of minerals of different specific gravity.

Allen Hazen has described[2] a method of sizing applied to the

[1] Private communication.

[2] Some Physical Properties of Sands and Gravels, with Special Reference to Their Use in Filtration. *Report of the Massachusetts State Board of Health*, 1892, pp. 539–555.

examination of sand used in filtration-plants. A beaker was used in which 5 grams of the fine material was agitated with water, allowed to settle 15 seconds and the water decanted. This was repeated by the addition of water three times, yielding a settled product which was dried and weighed, and the average size of grain determined by measurement with a microscope-micrometer. The amount of overflow-product was not weighed, but was calculated by difference.

Following this idea, experiments were tried by settling successively for 300, 120, 60, 30 and 15 seconds. Samples of the five settled products were examined under the microscope to determine their uniformity and sizes. These results were so gratifying, and the sizes so near those required for the Rittinger scale, that the periods of settling named were adopted and a series of tests carefully conducted. A beaker 70 mm. in diameter and 110 mm. deep was filled with distilled water to a depth of 90 mm., and 5 grams of ore, through a sieve with 0.069-mm. holes, was thoroughly stirred in. After standing quietly for 300 seconds, the water was quickly but carefully poured off without disturbing the settled material. The beaker was again filled to the 90-mm. mark and the process repeated. This was continued until the decanted water was clear. What remained in the beaker was similarly treated for periods of 120, 60, 30 and 15 seconds in succession. Distilled water was used at a temperature of about 20° Cent. (68° Fahr.), and was not allowed to vary beyond 18° Cent. and 22° Cent. (64.4° Fahr. and 71.6° Fahr.). Each product was allowed to settle, the clear water siphoned off, and the residues dried and weighed, except the finest product, which was determined by difference. It was found, in order to make duplicate tests agree, that each period of settling (300, 120, 60 and 30 seconds) should be repeated the same number of times in the separate tests, and that it was not safe to rely on the visual standard of the observer as to the clearness of the last decanted water. Five settlings seem to give good results, and do not take an unreasonable length of time; and while this plan was not adopted except during the latter part of the tests recorded in this paper, it is recommended as the most satisfactory method.

Table II. gives an idea of the degree of correspondence in results which are to be expected in sizing by settling in a

beaker. Duplicate trials were made on the same sample, and the work done on different days.

TABLE II.—*Sizing by Settling in Beakers.*

	First Result.	Second Result.
	Per Cent.	Per Cent.
Settled in 15 seconds...	34.9	36.9
Settled in 30 seconds...	27.1	27.7
Settled in 60 seconds...	17.1	14.2
Settled in 120 seconds..	6.5	5.9
Settled in 300 seconds..	3.1	3.3
Overflow by difference.	11.3	12.0
Total.....	100.0	100.0

TABLE IIa.—*Sizing-Scale for Sizes Finer than the Finest Obtainable Screens.*

Kind of Mineral.	Quartz- or Gangue-Rock.	Pyrite or Chalcopyrite.	Galena.
Smallest Grains.	Diameter in Millimeters.	Diameter in Millimeters.	Diameter in Millimeters.
Settled in water in 15 seconds.....	0.047	0.038	0.036
Settled in water in 30 seconds.....	0.034	0.028	0.024
Settled in water in 60 seconds.....	0.025	0.022	0.018
Settled in water in 120 seconds...	0.019	0.015	0.015
Settled in water in 300 seconds...	0.012	0.010	0.006

The sizes for the series of settled products were determined on the assumption that if the material were mostly quartz, practically the same quantities would have been obtained by screening and settling, if the screen-holes had the same diameters as the smallest grains of quartz in each of the settled products. If the material were mostly pyrite or mostly galena, the settled products would correspond in quantity to what would be obtained by different screen-sizes determined by measuring the smallest grains of pyrite or galena. If a sample under examination were made up of a mixture of quartz and a heavy mineral, in which neither greatly predominated, the product would be sorted rather than sized, and it is possible to express only an approximation to results which would have been obtained from sifting, by averaging the sizes of the minerals according to the judgment of the observer.

In sifting, it is evident that the two lesser dimensions determine the ability of grains to pass a square or a round hole; whereas, when measuring the same grains under a microscope, they never present their shortest diameter for measurement. This is illustrated by some measurements of the largest grains in the 15-second settled products. These grains had all been through a pentagonal hole with average diameter of 0.069 mm., but one sample showed the average diameter of the largest grains of quartz to be 0.100 mm., and of galena, 0.092 mm. The other sample showed the average diameter of the largest quartz-grains to be 0.105 mm., and of the chalcopyrite 0.096 mm. In the first of these determinations the average largest dimension presented was 0.120 mm., and the average smallest dimension was 0.080 mm.

The justification of the plan of measuring the smallest grains, in order to determine sizes, rests partly in the fact that their sizes give results which harmonize with the data secured by sifting, and with them yield smooth curves in plotting. As no other plan seemed more practical, it was adopted.

The diameter of the smallest particles in each settled product was determined as follows: A sample was placed under a microscope with a micrometer-measuring attachment, and the length and width of the smallest particle in the field of the microscope was measured. This was done with twenty or more separate fields for each sample, and the average of all measures was considered to represent the smallest particle in the sample. Separate measurements were taken on quartz- or gangue-rock, pyrite or chalcopyrite, and galena. On quartz-products, samples were measured independently by two different observers, so as to eliminate the personal equation as far as possible. A summary of all these measurements appears in Table III.

The quartz-sizes are used in the sizing-scale, Table I., and in the tabulated records of tests, because the major part of each sample consisted of quartz- or gangue-rock. The sizes for chalcopyrite or pyrite and galena are given in Table II*a*. Chalcopyrite and pyrite are recorded together, because it was impossible to identify them in the samples so as to measure them separately.

TABLE III.—*Sizing by Settling in Beakers.*

Summary of All Results of Measurements to Determine Sizes of Products.					
	Time of Settling in Seconds.	Number of Independent Determinations of Size.	Mean Diameter of Smallest Grain in Millimeters.		
			Highest Result.	Lowest Result.	Average.
Quartz- or gangue-rock. Specific gravity = 2.6.	15	6	0.054	0.040	0.047
	30	3	0.037	0.033	0.034
	60	3	0.027	0.024	0.025
	120	3	0.021	0.018	0.019
	300	3	0.013	0.010	0.012
Pyrite or chalcopyrite. Specific gravity = from 4. to 5.	15	3	0.045	0.034	0.038
	30	2	0.029	0.028	0.028
	60	2	0.025	0.019	0.022
	120	2	0.015	0.014	0.015
	300	2	0.011	0.009	0.010
Galena. Specific gravity = 7.6.	15	3	0.040	0.033	0.036
	30	1	.....	.....	0.024
	60	1	.....	.....	0.018
	120	1	.....	.....	0.015
	300	1	.....	.....	0.006

Complete tabulated results of the sizing of the mill products are shown in Tables IV. to XI.

TABLE IV.—*Sizing-Tests of Trommel-Products from Mill 22.*

(From Richards' *Ore-Dressing.*)

	Mill Sizes.					
	Through 12 on 6 mm.		Through 6 on 3 mm.		Through 3 mm.	
Number on Fig. 4.	1.		2.		4.	
Through.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.
16.0 on 11.2 mm.	4.0	4.0	.....	.....	.....	.....
11.2 on 8.02 mm.	34.8	38.8	.....	.....	.....	.....
8.02 on 5.61 mm.	45.3	84.1	2.4	2.4	.....	.....
5.61 on 3.94 mm.	13.5	97.6	45.3	47.7	0.2	0.2
3.94 on 2.69 mm.	1.7	99.3	48.5	96.2	20.8	21.0
2.69 on 1.89 mm.	0.3	99.6	2.5	98.7	26.8	47.8
1.89 on 1.49 mm.	0.1	99.7	0.5	99.2	13.5	61.3
1.49 on 0.945 mm.	.....	.....	0.3	99.5	14.6	75.9
0.945 on 0.667 mm.	.....	.....	0.2	99.7	7.8	83.7
0.667 on 0.493 mm.(a)	.....	.....	0.1	99.8	5.1	88.8
0.493 on 0.371 mm.	.....	.....	.....	.....	1.4	90.2
0.371 on 0.270 mm.	.....	.....	.....	.....	2.7	92.9
0.270 on 0.158 mm.	.....	.....	.....	.....	3.2	96.1
0.158 on 0.119 mm.	.....	.....	.....	.....	0.9	97.0
0.119 on 0.073 mm.	.....	.....	.....	.....	1.2	98.2
0.073 on 0.069 mm.	.....	.....	.....	.....	0.2	98.4
Fines.....	0.4	.....	0.3	.....	1.3	.....
Total.....	100.1	.....	100.1	.....	99.7	.....

(a) Round-hole sieves were used down to and including 0.493-mm. size, and square holes for finer sizes.

TABLE V.—*Sizing-Tests of Trommel-Products from Mill* 28.

(From Richards' *Ore-Dressing.*)

	Mill Sizes.																	
	Feed.		Through 40 on 25 mm.		Through 25 on 16 mm.		Through 16 on 12 mm.		Through 12 on 8 mm.		Through 8 on 5 mm.		Through 5 on 3.5 mm.		Through 3.5 on 2 mm.		Through 2 mm.	
Number on Fig. 5.	1.		2.		3.		4.		5.		6.		7.		8.		9.	
Through.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.
44.3 on 31.9 mm.	5.9	5.9	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
..9 on 22.3 mm.	16.6	22.5	100.0	100	17.9	17.9	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
2.3 on 16.0 mm.	13.3	35.8	.....	.....	82.1	100	4.5	4.5	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
16.0 on 11.2 mm.	16.4	52.2	.....	.....	.....	.....	94.9	99.4	62.4	62.4	.....	.....	.....	.....	.....	.....	.....	.....
11.2 on 8.02 mm.	9.8	62.0	.....	.....	.....	.....	0.6	100	36.0	98.4	.....	.....	.....	.....	.....	.....	.....	.....
8.02 on 5.61 mm.	8.8	70.8	.....	.....	.....	.....	.....	.....	1.6	100	58.9	58.9	0.3	0.3	.....	.....	.....	.....
5.61 on 3.94 mm.	6.2	77.0	.....	.....	.....	.....	.....	.....	.....	.....	40.4	99.3	72.0	72.3	.....	.....	.....	.....
3.94 on 2.69 mm.	4.7	81.7	.....	.....	.....	.....	.....	.....	.....	.....	0.5	99.8	27.7	100	51.0	51.0	.....	.....
2.69 on 1.89 mm.	2.9	84.6	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	44.2	95.2	.....	.....
1.89 on 1.49 mm.	1.7	86.3	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	4.3	99.5	5.6	5.6
1.49 on 0.945 mm.	2.2	88.5	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	21.2	26.8
0.945 on 0.667 mm.	1.6	90.1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	16.3	43.1
0.667 on 0.493 mm.(a)	1.3	91.4	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	13.7	56.8
0.493 on 0.371 mm.	0.4	91.8	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	5.6	62.4
0.371 on 0.270 mm.	1.0	92.8	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	8.9	71.3
0.270 on 0.158 mm.	1.5	94.3	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	13.8	85.1
0.158 on 0.119 mm.	0.8	95.1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3.8	88.9
0.119 on 0.078 mm.	1.2	96.3	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	5.6	94.5
0.078 on 0.069 mm.	0.3	96.6	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.9	95.4
Fines.....	3.1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	4.4	.....
Total.....	99.7	.....	100.0	.....	100.0	.....	100.0	.....	100.0	.....	99.8	.....	100.0	.....	99.5	.....	99.8	.....

(a) Round-hole sieves were used down to and including 0.493-mm. size, and square holes for finer sizes.

TABLE VI.—*Sizing-Tests of Trommel-Products from Mill* 30.

(From Richards' *Ore-Dressing.*)

	Mill Sizes.											
	Through 25 on 15 mm.		Through 15 on 10 mm.		Through 10 on 7 mm.		Through 7 on 5 mm.		Through 5 on 3 mm.		Through 3 mm.	
Number on Fig. 6.	1.		2.		3.		4.		5.		6.	
Through.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.
31.9 on 22.3 mm.	2.3	2.3	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
22.3 on 16.0 mm.	23.0	25.3	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
16.0 on 11.2 mm.	53.1	78.4	18.2	18.2	.....	.....	.....	.....	.....	.....	.....	.....
11.2 on 8.02 mm.	17.0	95.4	51.6	69.8	7.4	7.4	.....	.....	.....	.....	.....	.....
8.02 on 5.61 mm.	2.9	98.3	23.3	93.1	47.7	55.1	4.0	4.0	.....	.....	.....	.....
5.61 on 3.94 mm,	0.5	98.8	3.7	96.8	32.4	87.5	36.5	40.5	2.6	2.6	.....	.....
3.94 on 2.69 mm.	0.2	99.0	1.2	98.0	9.0	96.5	40.2	80.7	23.0	25.6	0.5	0.5
2.69 on 1.89 mm.	0.1	99.1	0.7	98.7	2.0	98.5	13.5	94.2	28.0	53.6	6.4	6.9
1.89 on 1.49 mm.	0.1	99.2	0.2	98.9	0.5	99.0	2.9	97.1	16.1	69.7	9.4	16.3
1.49 on 0.945 mm.	.....	.....	0.2	99.1	0.3	99.3	1.7	98.8	19.1	88.8	20.5	36.8
0.945 on 0.667 mm.	.....	.....	.....	.....	0.1	99.4	0.4	99.2	7.0	95.8	14.2	51.0
0.667 on 0.493 mm.(a)	.....	.....	.....	.....	.....	.....	0.1	99.3	2.3	98.1	11.3	62.3
0.493 on 0.371 mm.	.....	.....	.....	.....	.....	.....	.....	.....	0.4	98.5	2.6	64.9
0.371 on 0.270 mm.	.....	.....	.....	.....	.....	.....	.....	.....	0.5	99.0	6.0	70.9
0.270 on 0.158 mm.	.....	.....	.....	.....	.....	.....	.....	.....	0.4	99.4	9.7	80.6
0.158 on 0.119 mm.	.....	.....	.....	.....	.....	.....	.....	.....	0.1	99.5	3.3	83.9
0.119 on 0.073 mm.	.....	.....	.....	.....	.....	.....	.....	.....	0.2	99.7	4.3	88.2
0.073 on 0.069 mm.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.9	89.1
Fines.....	0.9	.....	0.9	.....	0.5	.....	0.5	.....	0.3	.....	9.5	.....
Total.....	100.1	.....	100.0	.....	99.9	.....	99.8	.....	100.0	.....	98.6	.....

(*a*) Round-hole sieves were used down to and including 0.493-mm. size, and square holes for finer sizes.

TABLE VII.—*Sizing-Tests of Trommel-Products from Mill* 38.
(From Richards' *Ore-Dressing.*)

	Mill Sizes.							
	Through 38.1 on 22.2 mm.		Through 22.2 on 9.5 mm.		Through 9.5 on 5 mm.		Through 5 on 2.5 mm.	
Number on Fig. 7.	1.		2.		3.		4.	
Through.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.
44.3 on 31.9 mm.	24.3	24.3	.....	.....	.....	.....	.....	.....
31.9 on 22.3 mm.	41.2	65.5	0.4	0.4	.....	.....	.....	.....
22.3 on 16.0 mm.	29.4	94.9	17.6	18.0	.....	.....	.....	.....
16.0 on 11.2 mm.	4.2	99.1	38.2	56.2	.....	.....	.....	.....
11.2 on 8.02 mm.	0.2	99.3	26.0	82.2	0.8	0.8	.....	.....
8.02 on 5.61 mm.	.....	.....	15.4	97.6	42.0	42.8	2.2	2.2
5.61 on 3.94 mm.	.....	.....	1.6	99.2	38.0	80.8	21.4	23.6
3.94 on 2.69 mm.	.....	.....	0.3	99.5	10.1	90.9	42.3	65.9
2.69 on 1.89 mm.	.....	.....	0.1	99.6	2.8	93.7	21.7	87.6
1.89 on 1.49 mm.	.....	.....	.....	.....	1.1	94.8	6.9	94.5
1.49 on 0.945 mm.	.....	.....	.....	.....	1.2	96.0	3.8	98.3
0.945 on 0.667 mm.	.....	.....	.....	.....	0.8	96.8	0.8	99.1
0.667 on 0.493 mm.(a)	.....	.....	.....	.....	0.7	97.5	0.3	99.4
0.493 on 0.371 mm.	.....	.....	.....	.....	0.2	97.7	.....	.....
0.371 on 0.270 mm.	.....	.....	.....	.....	0.5	98.2	.....	.....
0.270 on 0.158 mm.	.....	.....	.....	.....	0.7	98.9	.....	.....
Fines.....	0.7	.....	0.4	.....	0.8	.....	0.6	.....
Total.....	100.0	.....	100.0	.....	99.7	.....	100.0	.....

(a) Round-hole sieves were used down to and including 0.493-mm. size, and square holes for finer sizes.

TABLE VIII.—*Sizing-Tests of Classifier Products in Mill 22.*
(From Richards' *Ore-Dressing.*)

	No. 1 Hydraulic Classifier.								No. 1 Surface Current Box Classifier.					
	Feed.		First Spigot.		Second Spigot.		Over-flow.		First Spigot.		Second Spigot.		Over-flow.	
Number on Fig. 8.	4.		5.		6.		7.		8.		9.		10.	
Through.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.
5.61 on 3.94 mm. (*a*).	0.2	0.2	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
3.94 on 2.69 mm.....	20.8	21.0	2.4	2.4	1.0	1.0	.....	.....	.....	.....	.....	.....	.....	.....
2.69 on 1.89 mm.....	26.8	47.8	7.2	9.6	2.9	3.9	.....	.....	.....	.....	.....	.....	.....	.....
1.89 on 1.49 mm.....	13.5	61.3	6.1	15.7	2.5	6.4	.....	.....	.....	.....	.....	.....	.....	.....
1.49 on 0.945 mm.....	14.6	75.9	13.1	28.8	7.1	13.5	.....	.....	0.3	0.3	.....	.....	.....	.....
0.945 on 0.667 mm.....	7.8	83.7	14.2	43.0	8.2	21.7	0.3	0.3	0.5	0.8	.....	.....	.....	.....
0.667 on 0.493 mm. (*a*)..	5.1	88.8	17.0	60.0	10.3	32.0	0.6	0.9	1.5	2.3	.....	.....	.....	.....
0.493 on 0.371 mm.....	1.4	90.2	2.9	62.9	1.8	33.8	0.6	1.5	1.0	3.3	.....	.....	.....	.....
0.371 on 0.270 mm.....	2.7	92.9	12.4	75.3	10.8	44.1	2.0	3.5	4.8	8.1	.....	.....	.....	.....
0.270 on 0.158 mm.....	3.2	96.1	15.8	91.1	17.9	62.0	9.7	13.2	22.0	30.1	0.4	0.4	.....	.....
0.158 on 0.119 mm.....	0.9	97.0	3.3	94.4	8.2	70.2	5.7	18.9	13.2	43.3	1.3	1.7	0.4	0.4
0.119 on 0.073 mm.....	1.2	98.2	3.3	97.7	14.7	84.9	13.8	32.7	22.5	65.8	10.9	12.6	1.6	2.0
0.073 on 0.069 mm. (*a*)..	0.2	98.4	0.5	98.2	2.4	87.3	1.6	34.3	4.2	70.0	2.7	15.2	0.4	2.4
0.069 on 0.047 mm.....							6.3	40.6	18.0	88.0	13.6	28.9	6.7	9.1
0.047 on 0.034 mm.....							15.1	55.7	7.7	90.7	23.2	52.1	14.4	23.5
0.034 on 0.025 mm.....	1.3	.....	1.2	.....	11.7	.....	12.2	67.9	3.5	94.2	25.9	78.0	18.9	42.4
0.025 on 0.019 mm.....							7.7	75.6	1.3	95.5	6.7	84.7	12.8	55.2
0.019 on 0.012 mm.....							5.4	81.0	0.6	96.1	3.3	88.0	7.4	62.6
0.012 mm.....							18.6	.....	2.6	.....	11.6	.....	37.1	.....
Total.....	99.7	.....	99.4	.....	99.0	.....	99.6	.....	98.7	.....	99.6	.....	99.7	.....

(*a*) Round-hole sieves were used down to and including 0.493 mm.; then square holes down to and including 0.069 mm. Below 0.069 mm. settling in water was used, and the sizes given are diameters of quartz which settled a depth of 90 mm. in 15, 30, 60, 120 and 300 seconds. All of the settled products contained some grains of mineral heavier than quartz, and these grains were smaller than the quartz-grains.

TABLE IX.—*Sizing-Tests on Classifier Products in Mill* 28.

(From Richards' *Ore-Dressing.*)

	No. 1 Hydraulic Classifier.										No. 1 Surface Current Box Classifier.				(a).	
	Feed.		First Spigot.		Second Spigot.		Third Spigot.		Fourth Spigot.		First Spigot.		Second Spigot.		Spigot.	
Number on Fig. 9.	9.		10.		11.		12.		13.		14.		15.		16.	
Through.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.
2.89 on 1.89 mm. (b)..	.....	.....	0.3	0.3	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1.89 on 1.49 mm.....	5.6	5.6	28.5	28.8	0.1	0.1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1.49 on 0.945 mm.....	21.2	26.8	67.1	95.9	54.0	54.1	36.4	36.4	3.1	3.1	.....	.....	.....	.....	.....	.....
0.945 on 0.667 mm.....	16.3	43.1	3.1	99.0	36.3	90.4	43.3	79.7	30.8	33.9	0.04	0.04	.....	.....	.....	.....
0.667 on 0.493 mm. (b)..	13.7	56.8	0.3	99.3	7.7	98.1	10.7	90.4	32.5	66.4	0.1	0.1	.....	.....	.....	.....
0.493 on 0.371 mm.....	5.6	62.4	.....	.....	0.9	99.0	3.0	93.4	12.1	78.5	0.1	0.2	0.1	0.1	.....	.....
0.371 on 0.270 mm.....	8.9	71.3	.....	.....	0.7	99.7	3.0	96.4	13.2	91.7	0.5	0.7	0.2	0.3	.....	.....
0.270 on 0.158 mm.....	13.8	85.1	.....	.....	0.2	99.9	2.4	98.8	7.7	99.4	3.2	3.9	0.9	1.2	0.08	0.1
0.158 on 0.119 mm.....	3.8	88.9	.....	.....	.....	.....	0.5	99.3	0.2	99.6	3.0	6.9	0.7	1.9	0.1	0.2
0.119 on 0.073 mm.....	5.6	94.5	.....	.....	.....	.....	0.4	99.7	0.2	99.8	21.1	28.0	15.4	17.3	0.9	1.1
0.073 on 0.069 mm. (b)..	0.9	95.4	.....	.....	.....	.....	.....	.....	.....	.....	6.8	34.8	6.5	23.8	0.8	1.9
0.069 on 0.047 mm....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	52.8	87.6	65.5	89.3	20.5	22.4
0.047 on 0.034 mm....	.....	.....	.....	.....	0.1	.....	.....	.....	.....	.....	8.0	95.6	7.1	96.4	34.1	56.5
0.034 on 0.025 mm....	4.4	.....	.....	.....	.....	.....	0.3	.....	0.2	.....	1.8	97.4	1.9	98.3	23.2	79.7
0.025 on 0.019 mm....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.3	97.7	0.5	98.8	9.8	89.5
0.019 on 0.012 mm....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.2	97.9	1.4	.....	3.8	93.3
0.012 mm.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2.0	.....	.....	.....	5.6	.....
Total.....	99.8	.....	99.3	.....	100.0	.....	100.0	.....	100.0	.....	99.94	.....	100.2	.....	98.88	.....

(a) No. 1 whole current box classifier.

(b) Round-hole sieves were used down to and including 0.493 mm.; then square holes down to and including 0.069 mm. Below 0.069 mm. settling in water was used, and the sizes given are diameters of quartz which settled a depth of 90 mm. in 15, 30, 60, 120 and 300 seconds. All of the settled products contained some grains of mineral heavier than quartz, and these grains were smaller than the quartz-grains.

TABLE X.—*Sizing-Tests on Classifier Products in Mill* 30.
(From Richards' *Ore-Dressing.*)

	No. 1 Hydraulic Classifier.								Whole Current Box Classifier.					
	Feed.		First Spigot.		Second Spigot.		Third Spigot.		First Spigot.		Second Spigot.		Overflow.	
Number on Fig. 10.	6.		7.		8.		9.		10.		11.		12.	
Through.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.
5.61 on 3.94 mm. (*a*)	.....	.....	0.1	0.1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
3.94 on 2.69 mm	0.5	0.5	1.1	1.2	0.5	0.5	.....	.....	.....	.....	.....	.....	.....	.....
2.69 on 1.89 mm	6.4	6.9	14.2	15.4	6.7	7.2	0.2	0.2	.....	.....	.....	.....	.....	.....
1.89 on 1.49 mm	9.4	16.3	17.4	32.8	10.5	17.7	0.8	1.0	.....	.....	.....	.....	.....	.....
1.49 on 0.945 mm	20.5	36.8	29.2	62.0	24.4	42.1	3.8	4.8	.....	.....	0.3	0.3	.....	.....
0.945 on 0.667 mm	14.2	51.0	16.3	78.3	21.4	63.5	10.3	15.1	0.6	(*b*)	1.1	1.4	.....	.....
0.667 on 0.493 mm. (*a*)	11.3	62.3	10.1	88.4	17.5	81.0	23.0	38.1	0.3	0.3	2.5	3.9	.....	.....
0.493 on 0.371 mm	2.6	64.9	2.1	90.5	2.7	83.7	5.6	43.7	0.5	0.8	2.7	6.6	0.4	(*b*)
0.371 on 0.270 mm	6.0	70.9	3.8	94.3	7.0	90.7	16.2	59.9	1.5	2.3	7.3	13.9	0.2	0.2
0.270 on 0.158 mm	9.7	80.6	3.7	98.0	6.5	97.2	25.8	85.7	8.2	10.5	26.3	40.2	1.3	1.5
0.158 on 0.119 mm	3.3	83.9	0.7	98.7	1.2	98.4	6.2	91.9	4.1	14.6	13.0	53.2	1.9	3.4
0.119 on 0.073 mm	4.3	88.2	0.6	99.3	0.8	99.2	5.1	97.0	9.4	24.0	19.3	72.5	10.3	13.7
0.073 on 0.069 mm. (*a*)	0.9	89.1	0.1	99.4	0.1	99.3	0.6	97.6	1.5	25.5	4.0	76.5	2.5	16.2
0.069 on 0.047 mm	9.5	.....	0.4	.....	0.8	.....	1.7	.....	11.5	37.0	9.8	86.3	17.7	33.9
0.047 on 0.034 mm									12.3	49.3	6.2	92.5	27.1	61.0
0.034 on 0.025 mm									11.4	60.7	2.4	94.9	17.1	78.1
0.025 on 0.019 mm									8.2	68.9	1.1	96.0	6.5	84.6
0.019 on 0.012 mm									6.4	75.3	0.6	96.6	3.1	87.7
0.012 mm									23.6	.....	2.4	.....	11.2	.....
Total	98.6	.....	99.8	.....	99.6	.....	99.3	.....	99.5	.....	99.0	.....	99.3	.....

(*a*) Round-hole sieves were used down to and including 0.493 mm.; then square holes down to and including 0.069 mm. Below 0.069 mm. settling in water was used, and the sizes given are diameters of quartz which settled a depth of 90 mm. in 15, 30, 60, 120 and 300 seconds. All of the settled products contained some grains of mineral heavier than quartz, and these grains were smaller than the quartz-grains.

(*b*) This was all foreign material, such as chips, etc.

TABLE XI.—*Sizing-Tests of Classifier Products in Mill* 38.

(From Richards' *Ore-Dressing.*)

	No. 1 Unwatering-Box.		No. 1 Hydraulic Classifier.										No. 1 Whole Current Box Classifier.			
	Overflow.		First Spigot.		Second Spigot.		Third Spigot.		Fourth Spigot.		Overflow.		Spigot.		Overflow.	
Number in Fig. 11.	9.		5.		6.		7.		8.		10.		11.		12.	
Through.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.
3.94 on 2.69 mm. (*a*)	.....	.....	0.2	0.2	0.05	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
2.69 on 1.89 mm	.....	.....	6.4	6.6	1.9	1.9	0.4	0.4	0.04	0.04	.....	.....	.....	.....	.....	.....
1.89 on 1.49 mm	.....	.....	11.9	18.5	4.7	6.6	1.4	1.8	0.3	0.3	.....	.....	.....	.....	.....	.....
1.49 on 0.945 mm	.....	.....	26.3	44.8	16.4	23.0	7.2	9.0	2.1	2.4	.....	.....	.....	.....	.....	.....
0.945 on 0.667 mm	.....	.....	21.0	65.8	18.7	41.7	13.0	22.0	5.8	8.2	.....	.....	.....	.....	.....	.....
0.667 on 0.493 mm. (*a*)	.....	.....	17.1	82.9	21.3	63.0	22.4	44.4	14.8	23.0	.....	.....	.....	.....	.....	.....
0.493 on 0.371 mm	.....	.....	3.9	86.8	4.5	67.5	5.0	49.4	6.0	29.0	0.2	0.2	1.2	1.2	.....	.....
0.371 on 0.270 mm	0.1	0.1	5.7	92.5	11.8	79.3	15.8	65.2	18.0	47.0	0.3	0.5	2.1	3.3	.....	.....
0.270 on 0.158 mm	1.0	1.1	5.5	98.0	14.6	93.9	23.8	89.0	33.0	80.0	4.7	5.2	12.9	16.2	.....	.....
0.158 on 0.119 mm	1.4	2.5	0.8	98.8	3.4	97.3	5.2	94.2	10.9	90.9	4.9	10.1	7.9	24.1	.....	.....
0.119 on 0.078 mm	5.3	7.8	0.5	99.3	1.8	99.1	4.6	98.8	7.4	98.3	15.8	25.9	13.7	37.8	.....	.....
0.078 on 0.069 mm. (*a*)	1.8	9.6	0.02	99.3	0.06	99.2	0.2	99.0	0.5	98.8	2.3	28.2	2.6	40.4	.....	.....
0.069 on 0.047 mm	12.4	22.0									12.9	41.1	18.7	54.1	0.7	0.7
0.047 on 0.034 mm	5.8	27.8									9.4	50.5	9.0	63.1	1.1	1.8
0.034 on 0.025 mm	6.0	33.8	0.2	.....	0.4	.....	0.8	.....	1.2	.....	6.6	57.1	6.1	69.2	7.3	9.1
0.025 on 0.019 mm	7.5	41.3									8.6	65.7	5.7	74.9	15.3	24.4
0.019 on 0.012 mm	13.6	54.9									8.7	74.4	6.0	80.9	27.5	51.9
0.012 mm	46.3	.....									25.4	.....	17.4	.....	48.1	.....
Total	101.2	.....	99.52	.....	99.63	.....	99.8	.....	100.04	.....	99.8	.....	98.3	.....	100.0	.....

(*a*) Round-hole sieves were used down to and including 0.493 mm.; then square holes down to and including 0.069 mm. Below 0.069 mm. settling in water was used, and the sizes given are diameters of quartz which settled a depth of 90 mm. in 15, 30, 60, 120 and 300 seconds. All of the settled products contained some grains of mineral heavier than quartz, and these grains were smaller than the quartz-grains.

Tabular and Graphic Representation of Sizing-Tests—In Tables IV. to XI. two columns of figures are given for each product. The numbers in the first column show the per cent. by weight of each size. The numbers in the second column, headed "Cumulative per cent.," show the total per cent. of the size and all coarser, and represent the percentage of the whole sample which would rest on the screen, were it used alone. The cumulative per cent. is particularly useful in the preparation of some of the graphic diagrams explained later.

A graphic method of representing sizing-tests, to be of value, should present the results in a form more readily interpreted and understood than the tabulated figures. The following six conditions are desirable in the graphic record:

1. The relative quantities between the testing-screens should be clearly shown.

2. The relative quantities between screens of any set of sizes other than that represented by the testing-screens should be apparent, and the actual quantities should be easy of determination by the use of a scale.

3. The entire range of screen-sizes should be capable of representation in one figure.

4. The predominating grain-size of the sample should be prominently apparent.

5. The form of the plot should be unaffected by, and independent of, the sizing-scale used in making the test, thus affording, if desired, a means of direct comparison with data secured by a different sizing-scale.

6. The method should be simple, and the figures easily understood.

A number of methods have been used by other writers, and some of them will be discussed.

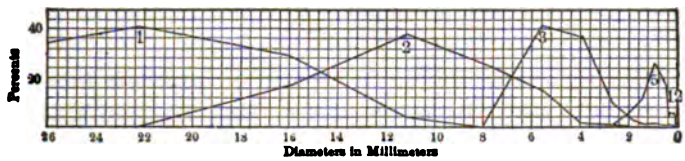
Prof. Courtney De Kalb, in his paper, Graphic Records of the Screening of Crushed Materials,[3] presents a series of drawings prepared by a method whereby the screen-sizes are plotted by measuring the diameters directly on a horizontal line, while the percentage of ore resting on each of the testing-screens is plotted vertically at the proper points.

Samples Nos. 1, 2, 3, 5 and 12 from Tables VII. and XI. are

[3] *Trans.*, xxviii., 468.

adapted to a discussion of the various methods of graphic representation, and Fig. 1 shows a set of five curves prepared by this method, yielding what may be called an ordinary direct plot. These curves fulfill none of the requirements for graphic records except the first and last; and these they fulfill no better than the tabulated figures. The omission of one of the sieves

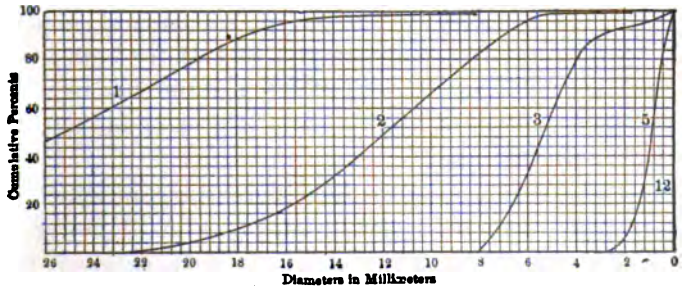
FIG. 1.



ORDINARY DIRECT PLOT OF SIZING-TESTS.

in making a test will greatly change the form of the curve, which is considerably dependent on the sizing-scale used. While the method is all right for some kinds of data, it is not suited to represent sizing-tests, and is positively misleading in some particulars: For example, curve No. 1 might be supposed to indicate that if a 20-mm. screen had followed the 22.3-mm. screen, it would have retained 37 per cent. of the ore; but that

FIG. 2.



CUMULATIVE DIRECT PLOT OF SIZING-TESTS.

is absurd, for the 16-mm. screen retained only 29.4 per cent., and the 20-mm. screen would, of course, fail to retain certain sizes that were caught by the 16-mm. screen. Other forms of plot show that a 20-mm. screen would have retained 12.5 per cent. of the ore, a fact which this curve affords no means of determination.

Luther Wagoner, in a paper, The Theory of Ore-Crushing,[4] uses a method wherein the scale of sizes is laid off on a horizontal line just as in the first described method, but the quantities are represented by indicating cumulative percentages on the vertical scale. The curves in Fig. 2 are drawn by this method, and this form of plot may be called a cumulative direct plot. Its use is illustrated by reference to curve No. 2, from which may be read that 18 per cent. of the material under test is coarser than 16-mm., 50 per cent. coarser than 12-mm., and 82 per cent. coarser than 8-mm. size.

This method of graphic representation answers most of the requirements, but it possesses one serious fault, in that it does not permit the representation of the entire range of sizes in one diagram. This is illustrated by Fig. 2, in which, at the scale adopted, curve No. 2 shows to good advantage, but this scale is too large to permit curve No. 1 to appear on the sheet without extending it as far again to the left, and at the same time, the scale is so small that curve No. 12 appears only as a vertical line. The cumulative direct plot recommends itself for its simplicity, and it will serve excellently for the study of data secured from the sizing of single samples in which the range of sizes does not exceed a ratio of 4 to 1 between diameters of coarsest and finest grains. Beyond these limits the curves are distorted to such an extent that the method is of little use.

The ratio of the openings in successive sizes of screens, as near as can be determined, is about 1 to 1.41 for both fine and coarse screens; but, of course, the actual difference of the area of openings represented by this ratio is greater for the coarse than for the fine screens.

It is desirable to have a method of plotting, in which equal distances on the plot represent equal, ratios of diameter, or, if we had been able to match the Rittinger scale precisely in getting the various sizing-screens, the data would be plotted at equal intervals on the horizontal scale, thus compressing the curve at the large end, and stretching the distances out on the small end. The same result is secured automatically by plotting the logarithms of the diameters on the horizontal line, and Fig. 3 represents this method applied to the same data

[4] *Transactions of the Technical Society of the Pacific Coast*, vol. iii., 1886, p. 45.

already presented by the other methods. If the logarithms of the diameters are multiplied by the constant 6.64, numbers very convenient for plotting are secured, because they are whole numbers for the diameters of the Rittinger scale, and these multiples of the common logarithms are given in Table I. for convenience in plotting. The negative quality of the logarithm of all numbers less than unity must be kept in mind, and since the logarithm of unity is zero, the original point for the scale of sizes will be at the unit diameter, and smaller sizes will be scaled to the right and larger sizes to the left from that point. This form of plot may be called the cumulative logarithmic plot. It has been used by the Massachusetts State

FIG. 3.

CUMULATIVE LOGARITHMIC PLOT OF SIZING-TESTS.

* The scale has been so chosen that the logarithm of any diameter must be multiplied by 6.64 in order to get the number of spaces on the plot.

Board of Health[5] for plotting sizing-curves of filtering-materials.

The curves resulting from this method fulfill every requirement for graphic representation. Its compliance with the first two requirements is easily shown by reference to curve No. 1 in Fig. 3. We read that 78 per cent. of the sample is coarser than a 20-mm. screen, and that 65.5 per cent. is coarser than a 22.3-mm. screen, therefore 12.5 per cent. of the material will fall between these sizes. It is also evident that the steep portions of the curve show the range of predominant sizes. Referring again to curve No. 1, we see that 95 per cent. of the sample falls between 16 mm. and 40 mm., and by curve

[5] Filtration of Sewage—Mechanical Composition of Materials Used. *Report of the Massachusetts State Board of Health*, 1891, pp. 428–430.

No. 5, it appears that 75 per cent. of the sample is quite uniformly distributed between 0.5 mm. and 2 mm.; also, that while some of the sample is probably as coarse as 2.8 mm.; also, 4 per cent. is coarser than 2 mm., and although some grains are as fine as 0.062 mm., only 20 per cent. is finer than 0.5 mm., and but 6 per cent. finer than 0.25 mm.

This method of graphic representation also affords a means of translating the records into tabular form to any scale of sizes which may be desired. The screens used in these tests conform only approximately to the Rittinger scale, but if it is desired, these results may be translated so that the tabular record will show the results which would have been obtained if screens conforming precisely to the Rittinger scale had been used. Such a translation of Table VII. has been made and is shown in Table XII.

TABLE XII.—*True Rittinger-Scale Sizes of Trommel-Products from Mill* 38.*

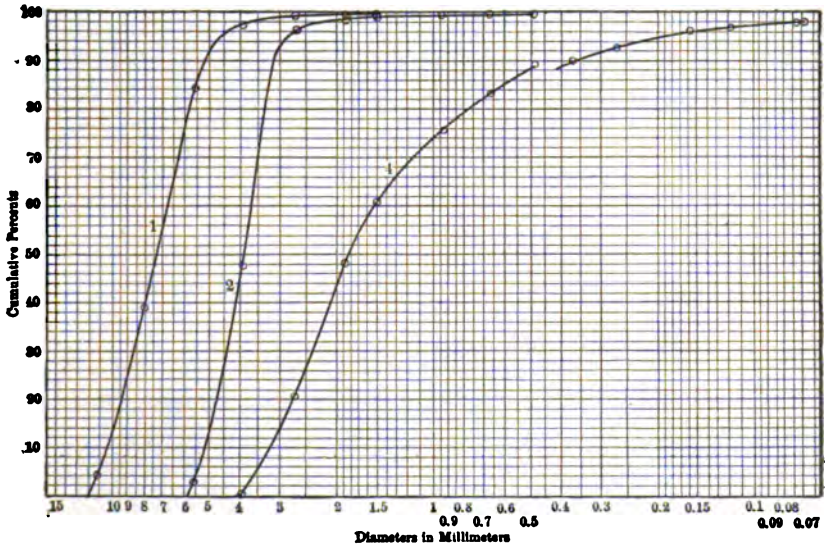
	Mill Sizes.							
	Through 88.1 on 22.2 mm.		Through 22.2 on 9.5 mm.		Through 9.5 on 5 mm.		Through 5 on 2.5 mm.	
Number on Fig. 7.	1.		2.		3.		4.	
Rittinger Sizes. Through.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.	Per Cent.	Cumulative Per Cent.
45.2 on 32.0 mm.	22.7	22.7	.....	.....	.....	.....	.....	.....
32.0 on 22.6 mm.	41.1	63.8	.....	.....	.....	.....	.....	.....
22.6 on 16.0 mm.	31.1	94.9	18.0	18.0	.....	.....	.....	.....
16.0 on 11.3 mm.	4.1	99.0	35.6	53.6	.....	.....	.....	.....
11.3 on 8.00 mm.	0.2	99.3	29.4	83.0	0.5	0.5	.....	.....
8.00 on 5.66 mm.	.....	.....	14.4	97.4	41.5	42.0	1.7	1.7
5.66 on 4.00 mm.	.....	.....	1.6	99.0	38.0	80.0	20.5	22.2
4.00 on 2.83 mm.	.....	.....	0.4	99.4	10.0	90.0	38.2	60.4
2.83 on 2.00 mm.	.....	.....	0.2	99.6	3.3	93.3	24.9	85.3
2.00 on 1.41 mm.	.....	.....	.....	.....	1.7	95.0	9.7	95.0
1.41 on 1.00 mm.	.....	.....	.....	.....	0.9	95.9	3.0	98.0
1.00 on 0.707 mm.	.....	.....	.....	.....	0.8	96.7	1.0	99.0
0.707 on 0.500 mm.	.....	.....	.....	.....	0.6	97.3	0.3	99.3
0.500 on 0.353 mm.	.....	.....	.....	.....	0.5	97.8	.....	.....
0.353 on 0.250 mm.	.....	.....	.....	.....	0.4	98.2	.....	.....
0.250 on 0.177 mm.	.....	.....	.....	.....	0.4	98.6	.....	.....
0.177 on 0.125 mm.	.....	.....	.....	.....	.....	.....	.....	.....
Fines.	0.7	.....	0.4	.....	1.4	.....	0.7	.....
Total.	99.9	.....	100.0	.....	100.0	.....	100.0	.....

* Derived from data shown in Table VII. and Fig. 7.

Tabular records are probably easier of comprehension to most readers than are the graphic diagrams, and it may be desirable to translate all sizing-records to one standard, thereby making possible, direct and easily understandable comparisons. Such a translation of any sizing-record is possible, provided the dimensions of the holes in the testing-screens are known.

The cumulative logarithmic method of graphic representation was used by Prof. Richards to represent all the tests recorded in this paper, and the plotted curves are reproduced in Figs. 4 to 11, inclusive. These curves differ from those in Fig. 3, used

FIG. 4.

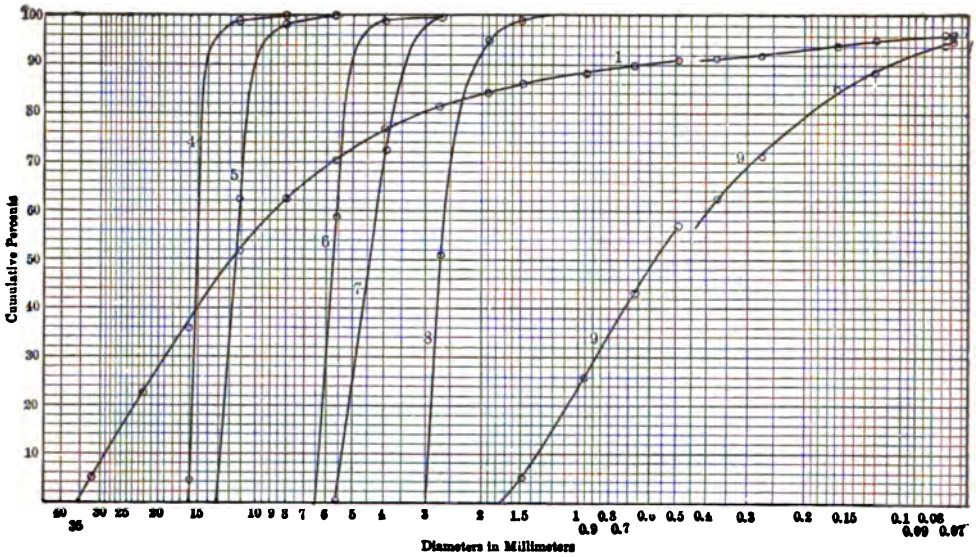


SIZING-TEST OF TROMMEL-PRODUCTS FROM MILL NO. 22. (See Table IV.)

for illustrating the method, in that they are plotted on logarithmic paper ruled for the purpose. Diameters in millimeters may be read directly from the horizontal scale. The original drawings showed the position of each size of the Rittinger scale by vertical colored lines, but they could not be reproduced in these engravings without confusion.

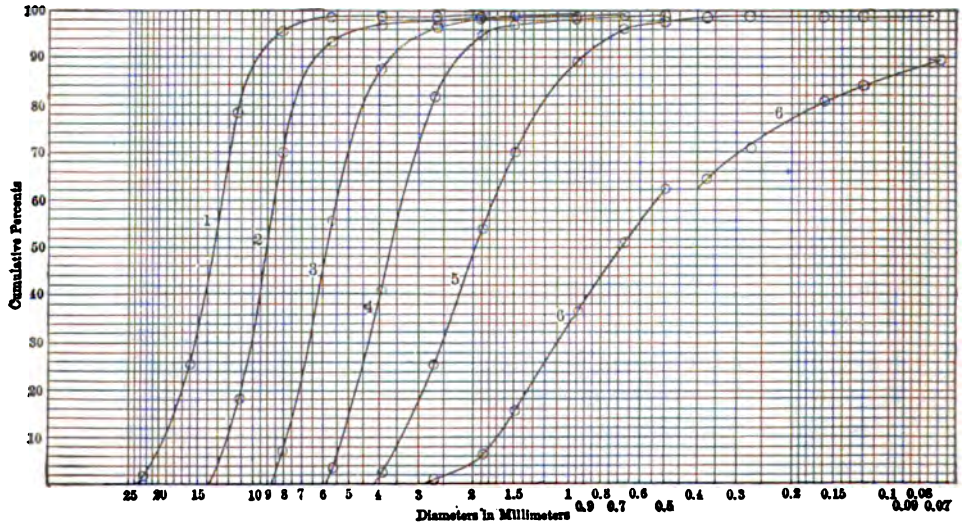
It will be noticed on Figs. 4 to 11, that for every sample containing material between 0.493- and 0.371-mm. size, there is a break in the curve at that point. In explanation of this, attention is directed to the fact already stated, that the 0.493-mm.

FIG. 5.



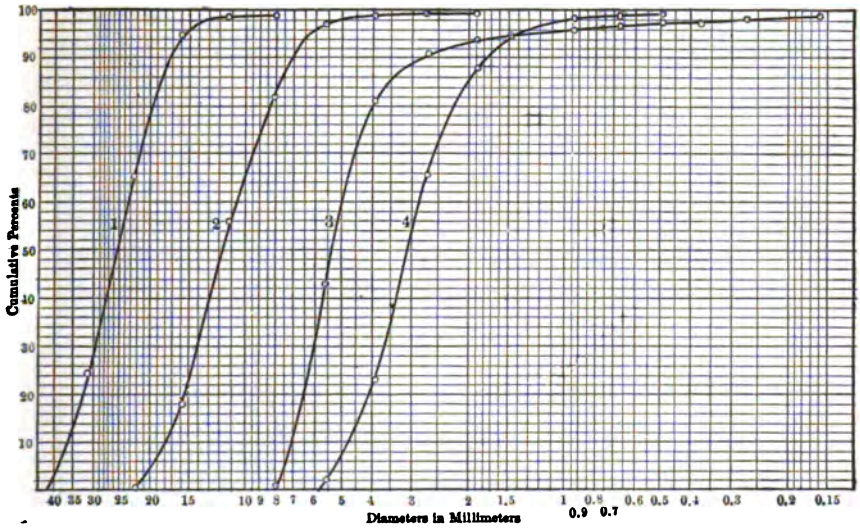
SIZING-TEST OF TROMMEL-PRODUCTS FROM MILL NO. 28. (See Table V.)

FIG. 6.



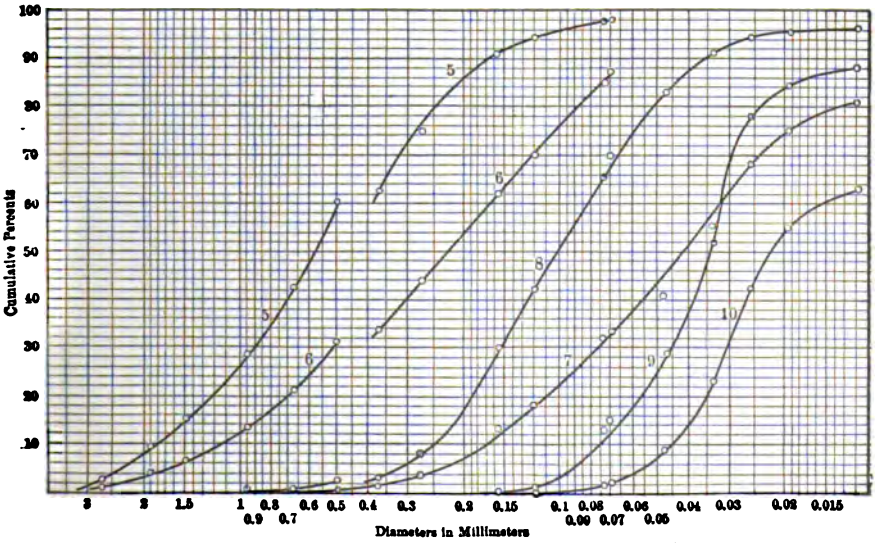
SIZING-TEST OF TROMMEL-PRODUCTS FROM MILL NO. 30. (See Table VI.)

FIG. 7.



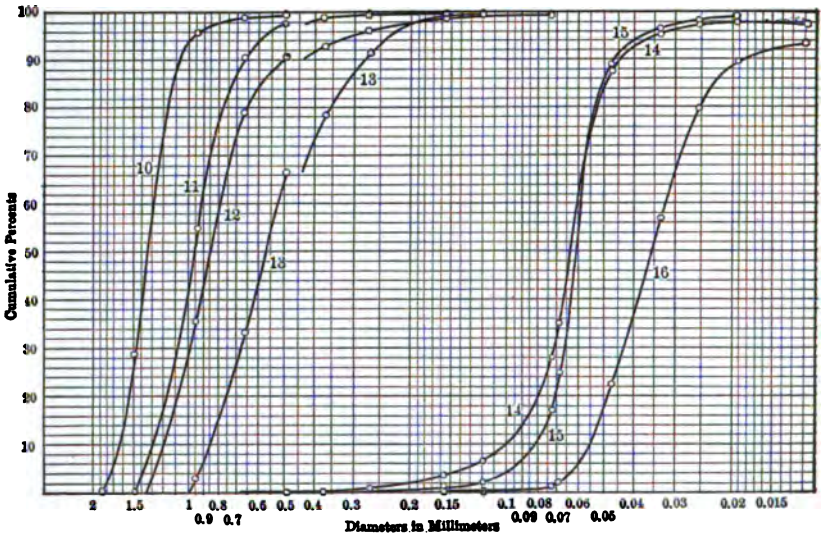
SIZING-TEST OF TROMMEL-PRODUCTS FROM MILL NO. 38. (See Table VII.)

FIG. 8.



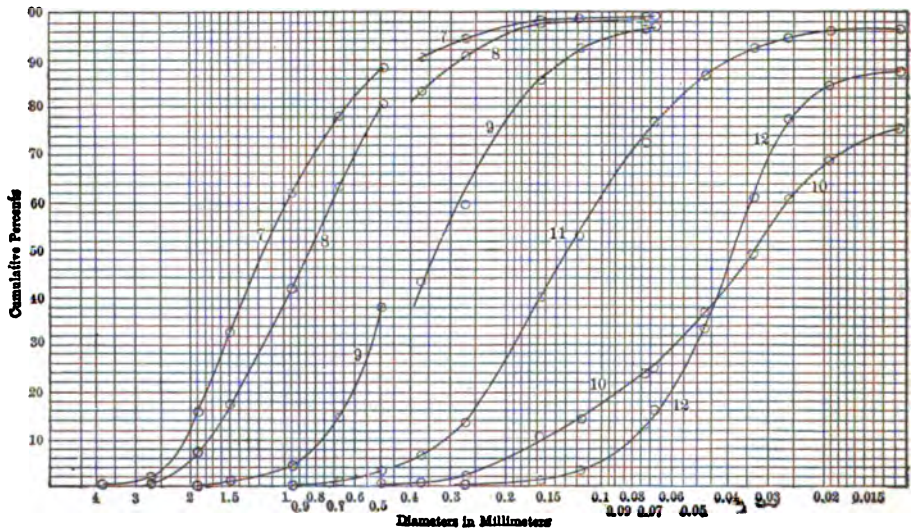
SIZING-TEST OF CLASSIFIER-PRODUCTS FROM MILL NO. 22. (See Table VIII.)

FIG. 9.



SIZING-TEST OF CLASSIFIER-PRODUCTS FROM MILL NO. 28. (See Table IX.)

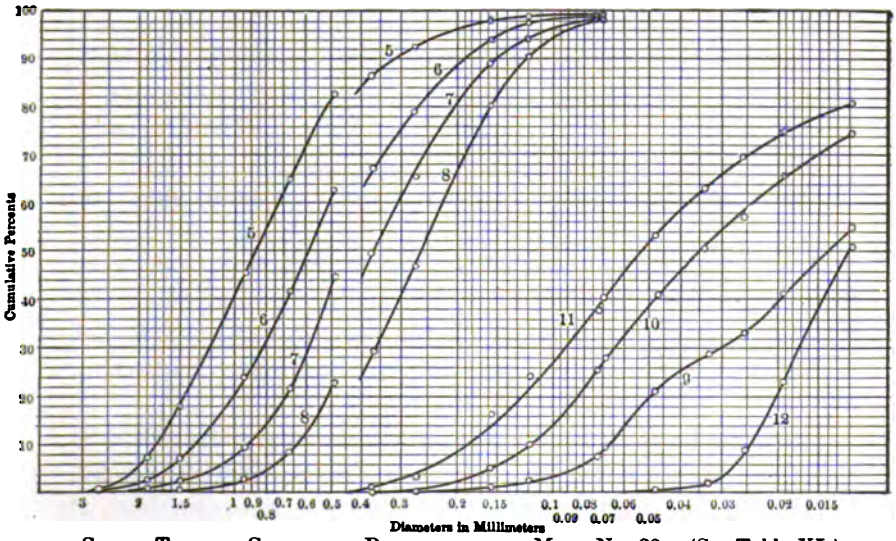
FIG. 10.



SIZING-TEST OF CLASSIFIER-PRODUCTS FROM MILL NO. 30. (See Table X.)

and all coarser screens have round holes, while the 0.371-mm. and all finer screens have square holes. A round-hole screen will retain finer material and, therefore, more material, than a square-hole screen of the same nominal size; consequently, if the fine screens could have been obtained with round holes, or if the square-hole screens had been used throughout, the curves would be continuous and unbroken, without suffering any alteration of form or direction. It is, therefore, believed to be better to sacrifice whatever advantage pertains to the use of round-hole screens, for those cases in which sizes smaller

FIG. 11.



SIZING-TEST OF CLASSIFIER-PRODUCTS FROM MILL No. 38. (See Table XI.)

than 0.5 mm. enter into the problem; and it is then recommended to use square-hole screens for all sizes.

The distance which either part of the curve must be moved, parallel to the scale of sizes, to bring the two curves together, will represent the practical difference in effective size, regarding screening-qualities, between round and square holes of the same nominal size. By this means, the diagrams indicate that a square hole, to retain just the same proportion of the sample as a round hole 0.500 mm. diameter, will measure from 0.406 to 0.435 mm., the first dimension being deduced for Mills Nos. 22 and 30, and the latter dimension for Mill No. 28.

This difference is probably due to some persistent difference in the shape of the grains of the mineral. Flat grains retained by a round hole will be readily passed by the diagonal dimension of a square hole of the same nominal size; and a difference in the proportion of flat and splinter-shaped grains will explain the difference noted. Such a difference in the shape of grains in different mills is not surprising, and might be due to the cleavage of the ore, or, possibly, in part, to some difference in the methods of crushing.

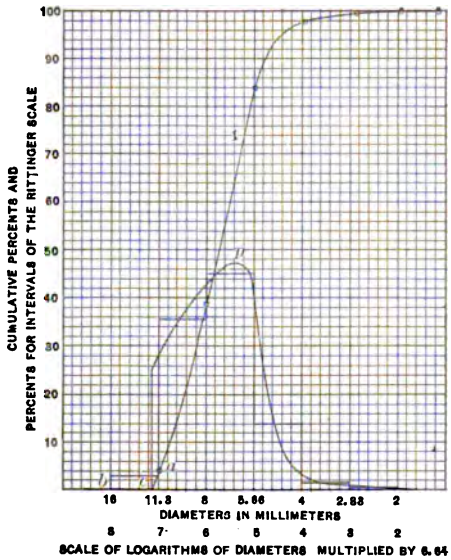
It will be noticed that no curves appear on Fig. 5 corresponding to Nos. 1 and 2 on Table V. These tests were on samples of very closely sized trommel-products, and the data furnish only one plotted point for No. 1, and but two points for No. 2, neither being sufficient to yield a curve. A more closely spaced sizing-scale, or intermediate screen-sizes, should have been used for these, also for tests Nos. 5, 6, and 8, Table V. and Fig. 5, in each of which more than 50 per cent. appears in the first size, yielding, therefore, no plotted point near the line of zero per cent.

Another method of graphic representation was considered and discussed in connection with these experiments; and while its nature does not recommend it for such general use as the cumulative direct plot, it has, nevertheless, some interesting and useful qualities.

Referring to Figs. 12 and 13, the rising curves are identical with curves No. 1, Fig. 4, and No. 5, Fig. 8, for Mill No. 22, except that the diameters are plotted by the same method as in Fig. 3, but on a larger scale. The heavy vertical section-lines correspond to the sizes of the Rittinger scale.

The steps in the curve are made by plotting on the vertical scale the percentages which would have rested on each sieve of a precise Rittinger scale. These figures are deduced by the method explained for Table XII.; and the height of each step, measured by the vertical scale on the left, represents the percentage, if sized by the true Rittinger scale. On the scale of logarithms, the horizontal length of each step is unity, and the area of the rectangle between two adjoining Rittinger sizes, bounded by the step and the base line, is equal to the percentage of that size. These rectangles represent in a pictorial way the relative distribution of sizes in the samples, where the ratio between successive sizes is uniform.

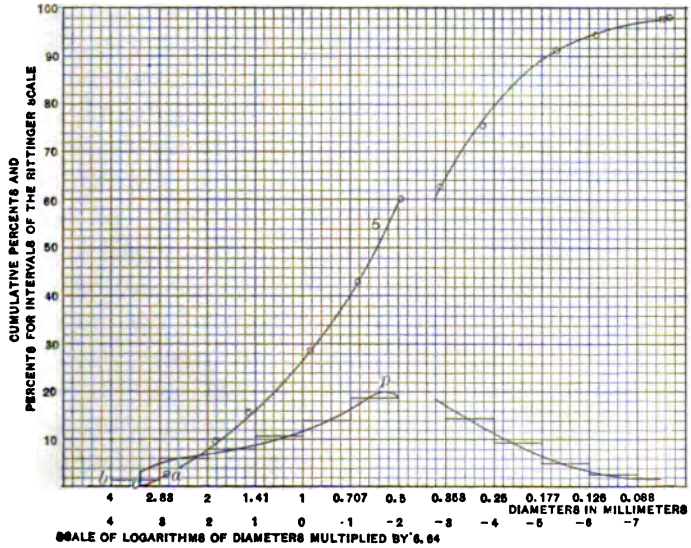
FIG. 12.



Each fifth vertical section line corresponds to a size of the Rittinger Scale.

CUMULATIVE AND SECONDARY LOGARITHMIC PLOTS FOR SAMPLE No. 1. MILL No. 22. (See Table IV. and Fig. 4.)

FIG. 13.



Each fifth vertical section line corresponds to a size of the Rittinger Scale.

CUMULATIVE AND SECONDARY LOGARITHMIC PLOTS FOR SAMPLE No. 5. MILL No. 22. (See Table VIII. and Fig. 8.)

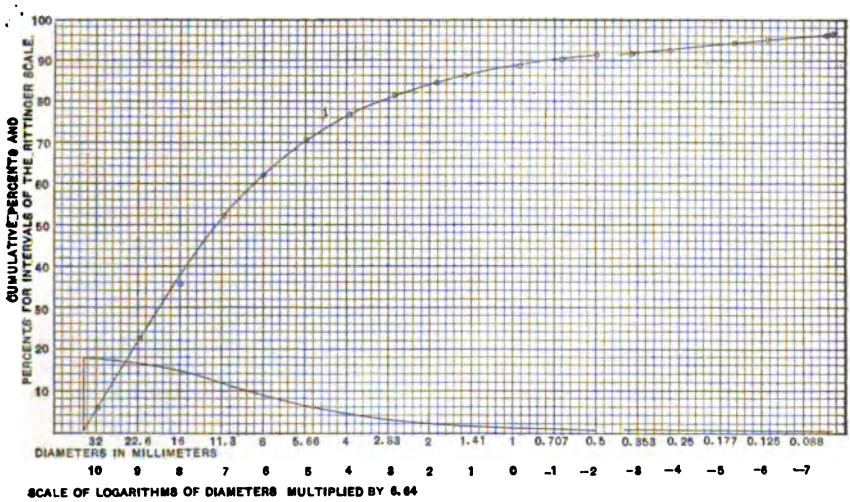
A comparison of this stepped figure, with the primary curves from which it is derived, shows a certain inconsistency at the starting-point on the left, where the primary curve leaves the base-line at an abrupt angle. If these two figures are to be consistent, either the primary curve should curl up at the lower end and extend to the point *b*, or the step at that point should not extend to the left beyond the point where the primary curve touches the base line at *c*. The data furnish no means of determining just how the primary curve should proceed from the nearest plotted point *a* to the base line. We have, however, on curve No. 4, Fig. 4; curve No. 7, Fig. 5; curves Nos. 2 and 3, Fig. 7; and curves Nos. 10 and 11, Fig. 9; examples in which plotted points of data appear close to the horizontal base-line, and in these, the data imperatively indicate that the curve should approach the base-line at a steep angle. No exceptions to this rule appear in samples of sized materials in which the maximum grain is coarser than 4-mm. size. These curves are, therefore, followed as a pattern in the construction of the primary curve in Figs. 12 and 13.

Assuming that both primary curves in Figs. 12 and 13 are properly terminated, any series of rectangles which will represent by their areas, the distribution of sizes, must begin at the point *c*, where the primary curve intersects the base-line. On this new basis, by taking horizontal distances less than before, measuring the difference between ordinates on the primary curve at the points selected and dividing by the horizontal distance of the scale of logarithms, a many-stepped figure will be obtained, in which the area of each narrow rectangle represents the percentage of material between the two sizes taken. If this idea be carried to infinitely small horizontal steps, a smooth curve will result, of which the ordinate at any point will be proportional to the tangent of the angle which the primary curve makes at the point with the horizontal base line. The two closed curves in Figs. 12 and 13, following pretty closely the stepped figures except at the point of beginning, is such a curve; which may be called a secondary logarithmic plot. The area enclosed by the curve and the base line, between any two vertical lines, represents the percentage of material in the sample, between the two sizes corresponding to the vertical lines taken. The scale is so chosen for Figs. 12 to 14

and 16 to 18, that 25 squares represent 10 per cent.; and the percentage between any two adjoining Rittinger sizes is shown by the average height of the curve between them, measured by the vertical scale of percentages on the left of the plot.

This curve shows pictorially the relative quantities between any set of sizes, but the actual quantity is not easy of determination except by measurement with a planimeter. It cannot, therefore, fill the place of the cumulative logarithmic plot as a simple method of graphic representation. The pictorial quality of this diagram is its chief merit, as it shows the predominating

FIG. 14.



Each fifth vertical section line corresponds to a size of the Rittinger Scale.

CUMULATIVE AND SECONDARY LOGARITHMIC PLOTS FOR SAMPLE NO. 1. MILL NO. 28. (See Table V. and Fig. 5.)

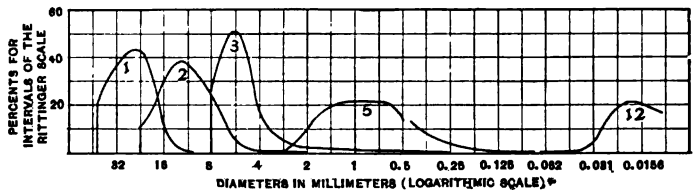
sizes by a glance at the vertical scale. Referring to Fig. 12, it is seen that no grains will be found in the sample coarser than the size represented by the point *c*. This point is at 7.15 on the scale of logarithms; divide this by 6.64 and we have 1.077, which is the common logarithm of the size; and a reference to a table of logarithms shows it to correspond to 12.0 mm.

Just as should be expected in a sized product, a considerable proportion of the coarsest grains is shown by the vertical height of the curve at this point. The predominating size is found, however, at the point *p*. The reading of the scale of

logarithms at this point is 5.4, which corresponds to a diameter of 6.5 mm. This information is yielded, likewise, by the primary curve, the point of maximum steepness indicating the predominating size, but it is not pictured so clearly to the eye as by the secondary curve. Moreover, the construction of the secondary curve is helpful to an understanding of the significance of the primary curve and it is suggested to use them together on the same plot.

Another plot on a different character of material, Fig. 14, is presented, which represents crushed ore, as it is fed to the sizing-trommels in Mill No. 28. It might be supposed, by an inspection of the tabular record, Table V., that the predominating size would be between 22.3 and 31.9 mm. or between 11.2 and 16.0 mm. Referring, however, to the secondary plot, it appears

FIG. 15.



* The scale has been so chosen that the logarithm of any diameter must be multiplied by 6.64 to get the number of spaces on the plot, and each vertical section line corresponds to a size of the Rittinger Scale. Areas enclosed by curves represent percentages and 1 square = 10 per cent.

SECONDARY LOGARITHMIC PLOT OF SIZING-TESTS.

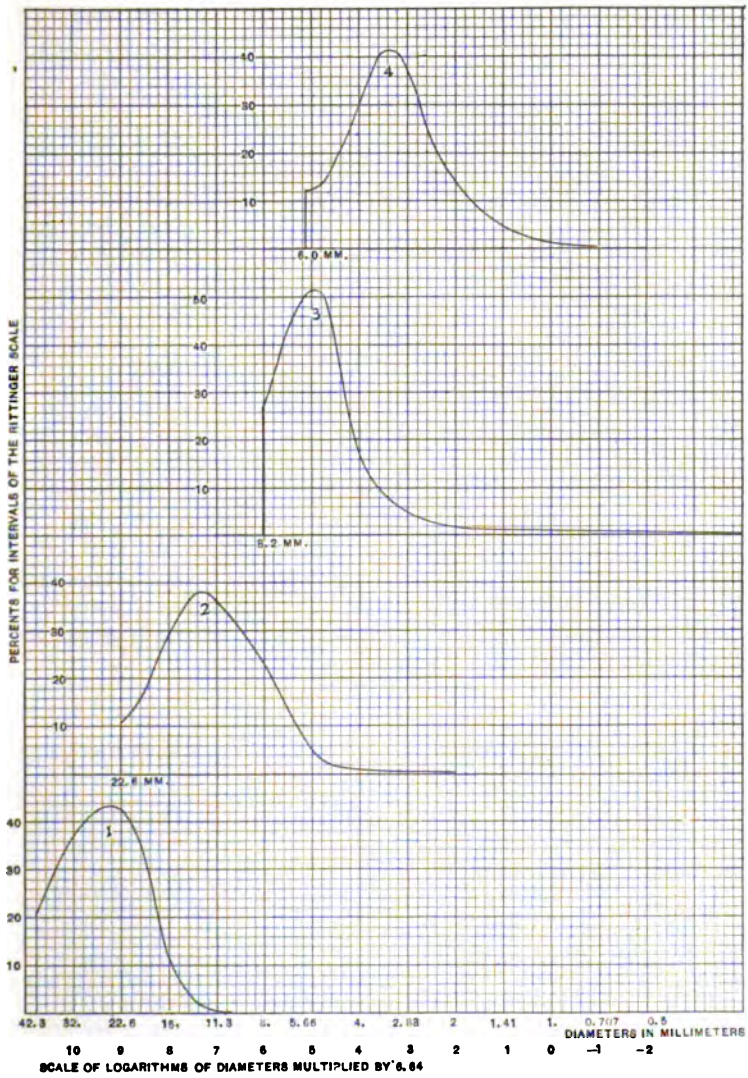
that the coarsest size predominates. This size corresponds to 10.32 on the scale of logarithms, which equals a diameter of 35.8 mm. From this size downwards, the proportion of material falling between any fixed ratio of sizes becomes progressively less and less.

Fig. 15 is a secondary logarithmic plot of the same data presented in Figs. 1, 2 and 3, prepared for the purpose of comparison. It is interesting to note that, while in Fig. 3, curve 12 seems only to have been begun, Fig. 15 shows that its culminating point has been passed, indicating that the size 0.017 mm. predominates.

Figs. 16 to 18 represent the data in Tables VII. and XI. plotted as secondary logarithmic curves, evolved from the cumulative logarithmic curves in Figs. 7 and 11. Curves Nos. 1,

2, 3 and 4, representing trommel-products, show similar characteristics, and while they overlap somewhat, they show, as should

FIG. 16.

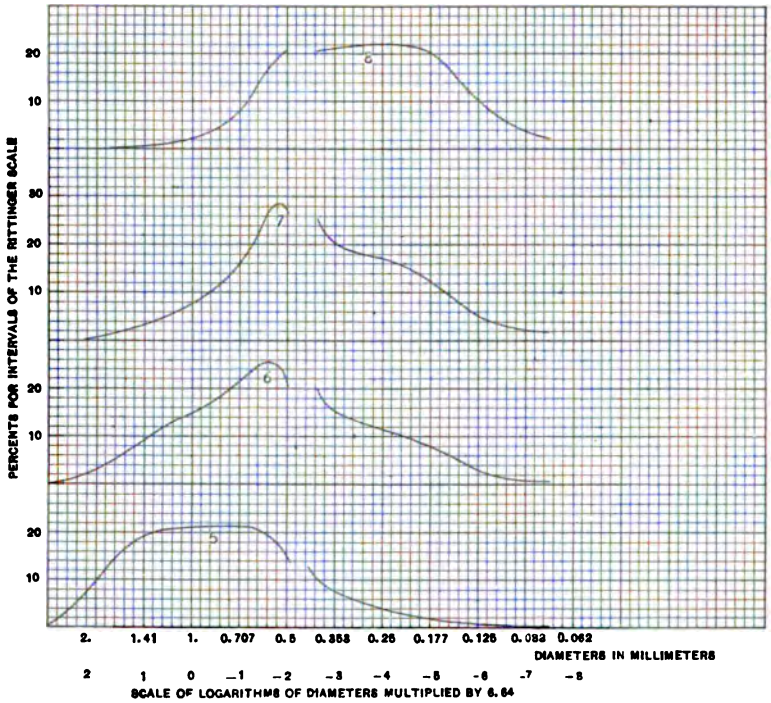


The vertical scale is so chosen that for areas enclosed by curves representing percents 25 squares = 10 per cent.

SECONDARY LOGARITHMIC PLOTS. SIZING-TEST OF TROMMEL-PRODUCTS FROM MILL No. 38. (See Table VII. and Fig. 7.)

be expected, a narrow range above and below the predominating size. Curves Nos. 5, 6, 7 and 8 (Fig. 17) represent

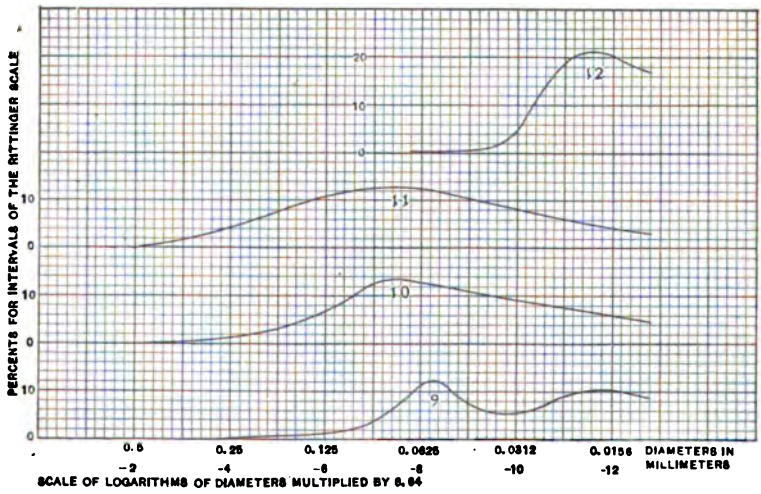
FIG. 17.



The vertical scale is so chosen that for areas enclosed by curves representing percentages, 25 squares = 10 per cent.

SECONDARY LOGARITHMIC PLOTS. SIZING-TEST OF CLASSIFIER-PRODUCTS FROM MILL No. 38. (See Table XI. and Fig. 11.)

FIG. 18.



The vertical scale is so chosen that for areas enclosed by curves representing percentages, 25 squares = 10 per cent.

SECONDARY LOGARITHMIC PLOTS. SIZING-TESTS OF CLASSIFIER-PRODUCTS FROM MILL No. 38. (See Table XI. and Fig. 11.)

spigot-products of the classifiers. No doubt there is some reason for the difference in form of these curves. Curves Nos. 5 and 8 present a flat top showing a wide range of predominating sizes, while curves Nos. 6 and 7 indicate a sharper definition of the predominating size. Probably the second and third spigots, represented by curves Nos. 6 and 7, are producing a more closely sorted product than the others. No. 9 (Fig. 18) represents the overflow of the No. 1 unwatering-box. It has a double hump, and is the only curve, prepared from the data presented, which shows any departure from what seems to be a type of secondary curve having one point of maximum height.

Conclusions.

Four methods of plotting sizing-tests have been described in this paper. The ordinary direct plot is dismissed as unsuited for sizing-diagrams. The cumulative direct plot is simple and fulfills most of the requirements tolerably well, but it does not permit the representation of the entire range of sizes on one diagram. The latter difficulty is overcome by the use of the cumulative logarithmic plot, which fulfills all requirements. The secondary logarithmic plot is evolved from the cumulative logarithmic plot and presents the data for pictorial comparison as none of the others does.

When a sample has only a limited range of sizes, and the plot is desired simply to determine, by interpolation, the exact quantities between any set of sizes other than those of the sizing-scale, a cumulative direct plot answers the purpose. For general analysis and comparative study of results of sizing-tests, the cumulative logarithmic plot, supplemented by the secondary logarithmic plot, is recommended as the most satisfactory.

This paper is a development of ideas suggested by the experiments made for Prof. Richards' new book, and in closing, I wish to express my indebtedness to Mr. C. D. Demond and Mr. C. E. Locke, assistants until lately to Prof. Richards, for suggestions and criticisms during the progress of the tests and the discussions of the data.

The Volcanic Origin of Oil.*

BY EUGENE COSTE, TORONTO, ONTARIO, CAN.

(Atlantic City Meeting, February, 1904.)

IN a recent paper[1] I took exception to the opening paragraph of Mr. Hill's paper, in which he says:—

"In endeavoring to interpret the geological occurrence of oil, the geologist is confronted by the fact that science has not yet solved the problem of its origin, which lies at the root of the subject. . . . For the present we must consider oil as a material in the rocks, the origin of which is still unexplained."

I take this occasion to repeat my objection to the above statement, as well as to the similar statement of Mr. C. W. Hayes and William Kennedy in their recent bulletin[2] on the same oil-fields:—"The origin of petroleum is one of the most obscure problems by which geologists are confronted."

I do not hesitate to say that these statements, which claim we know so little to-day of the origin of petroleum, are not warranted in the present state of geological science, and that, on the contrary, geology can to-day most clearly prove the origin of oil to be inorganic and the result of solfataric volcanic emanations.

It has been so long assumed that oil or bitumen is due to the decomposition of the organic remains of the sedimentary rocks, that the words organic and bituminous are used synonymously and, accordingly, without attempt at explanation, the origin of oil is in every case ascribed to some bituminous-shale horizon, more or less full of organic matter. Such falla-

* [SECRETARY'S NOTE.—A discussion of the paper of Robert T. Hill, entitled The Beaumont Oil-Field, with Notes on Other Oil-Fields of the Texas Region, which was prepared for the Franklin Institute and presented at a joint session of the Mining and Metallurgical section of that Society with this Institute, May 15, 1902. *Trans.*, xxxiii., 363. This discussion also is a joint contribution to both societies.]

[1] Volcanic Origin of Natural Gas and Petroleum, *Journal of the Canadian Mining Institute*, vol. vi., p. 73.

[2] Oil-Fields of the Texas-Louisiana Gulf Coastal Plain, *Bulletin U. S. Geological Survey*, No. 212, Washington, 1903.

cious reasoning, attributing the origin of oil to bituminous shales or to shales containing oil,—that is to say to oil,—of course, proves nothing except that the question is prejudged and entirely misunderstood by a great many geologists who seem to regard the abundant geological proofs that oil did not originate according to the above axiom, as extraordinary phenomena and profound mysteries not yet solved by science.

I wish to point out that this problem is pre-eminently a geological problem and not a chemical problem. Chemists can extract hydrocarbons from organic remains and can also produce hydrocarbons from simple mineral reactions (as in the commercial manufacture of acetylene, for instance), but the question before us is not, What are all the numerous chemical reactions capable of producing hydrocarbons or bitumens? but only, How does nature do it? What are the geological facts in this connection? If in chemistry hydrocarbons are generally grouped and classed among the organic compounds, that is not a proof that the natural hydrocarbons have geologically an organic origin, and that, geologically speaking, organic and bituminous are synonymous terms. On the contrary, the geological evidence is clear and strong, disproving the organic origin of bitumens, or hydrocarbons (excepting some marsh gas) found in the earth's strata, and showing plainly that the natural geological process of to-day, and of ages past, in the formation of these products, is a mineral or inorganic process. This geological evidence is so well known and so indisputable that a simple enumeration of it seems to me to be sufficient:—

1. Animal organic remains or bodies are never entombed in the rock-formations, and, therefore, cannot there produce oil or anything else.

2. Vegetable organic remains in the rock-formations always decompose into carbonaceous matter: That is, peat, lignite and coal, with a small production of marsh-gas which, however, either escapes in the atmosphere or remains in the coal, and has evidently nothing whatever to do with the natural gas and oil of the gas- and oil-fields.

3. Further distillation of carbonaceous matter has not taken place in nature, except locally, in very rare instances, as is proved by all the lignite- and coal-beds of the sedimentary strata.

4. As reviewed elsewhere,[3] and as able observers have repeatedly recorded, not only are gaseous, liquid and solid hydrocarbons or bitumens among the most important products of the solfataric volcanic emanations in the volcanic districts of the earth, but also carbonic acid, chlorides (mostly common salt), hydrogen sulphide, sulphur, gypsum, and hot calcareous and siliceous waters are always the remaining conspicuous products of these emanations; and all these associated products together, stamp the solfataric volcanic phenomena with a unique and unmistakable seal.

That this volcanic process is the normal and orderly mode of petroleum-production is to me a most clearly established geological fact, for the following reasons which I have also discussed at length in the paper already quoted:[4]

1. It is the only geological process of petroleum-production to be witnessed in active operation to-day, in nature.

2. In all the oil- and gas-fields or petroleum-deposits, the gaseous products are under a strong pressure which is not artesian or hydrostatic, which increases with depth, and which cannot be anything else but a volcanic pressure.

3. In some of the oil- and gas-fields, heated waters, oils and gases are met with.

4. All the oil- and gas-fields bear, imprinted largely through the products associated with the oil and gas, the seal referred to above as the distinct characteristic of solfataric emanations.

5. The oil- and gas-fields are located along the faulted and fissured zones of the crust of the earth, parallel to the great orogenic and volcanic dislocations.

6. Oil, gas and bitumens are never indigenous to the strata in which they are found—they are secondary products impregnating and cutting porous rocks of all ages, exactly as volcanic products alone can do.

7. Oil and gas are stored products, in great abundance in certain localities, while neighboring localities often are entirely barren; and many of the strata among which they are found are so impervious, that the source of these hydrocarbons must be the volcanic source below, which alone is abundant enough, and alone possesses sufficient energy, to force and accumulate

[3] *Journal of the Canadian Mining Institute*, vol. vi., p. 73.
[4] *Idem.*

such large quantities of these and associated products in so many spots through such impervious strata.

No other oil- or gas-fields conform to the foregoing statements more closely and more plainly than some of the fields under review in Mr. Hill's paper. Indeed Mr. Hill shows that the following conditions exist in the Texas-Louisiana Coast Prairie:

1. The oil and gas are always found under small mounds or salt islands which are recent gentle quaquaversal uplifts or uplifted domes. This is fully confirmed by C. W. Hayes[5] and by others.

2. The salt water and the oils under these mounds, in some cases, are still hot.

3. The oil and gas under these mounds do not occur in any definite stratum, but in spots of many strata, and in very large quantities in these limited areas.

4. The same may be said of the products associated with the oil or gas under these mounds, viz., sulphur, hydrogen sulphide, salt, gypsum, limestone and dolomite which form, under these mounds, many irregular masses and pockets without stratigraphical order of any kind.

5. The above associated products, like the oil and gas, are not found in the wells drilled outside of the mounds, except in very small quantity, while under the mounds they often form masses hundreds and thousands of feet in vertical thickness.[6]

If these well established facts are kept carefully in mind, no other conclusion can be reached but the one adopted by Mr. Hill, that "these materials (of the mounds associated with the oil and including the oil) have originated by secondary replacement, and that they may be of Post-Tertiary age."

But, in his explanation of these secondary replacements under the mounds, Mr. Hill certainly does not go far enough. In my opinion, the explanatory hypothesis which he offers[7] as a "basis for discussion," and not "as a final explanation," should be altered to read as follows:

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[5] Contributions to Economic Geology, 1902, *Bulletin U. S. Geological Survey*, No. 213, pp. 345, 347, 348.

[6] See Capt. Lucas' paper, Rock-Salt in Louisiana, *Trans.*, xxix., 462; also *Bulletin U. S. Geological Survey*, No. 212, p. 126.

[7] *Trans.*, xxxiii., 397 (p. 35 of the pamphlet edition).

"The oil-, sulphur-, salt-, natural-gas, and hydrogen-sulphide pockets of the Texas Coastal Plain are not indigenous to the strata in which they are found, but are the resultant products of columns of hot saline, siliceous, calcareous, magnesian, and sulphur water and vapors associated with sulphur and hydrocarbon gases and vapors which have ascended, under volcanic pressure, at points along lines of structural weakness, and have disseminated also in more or less minute quantities a little oil, gas, sulphur and salt through thousands of feet of shales, sand and marine littoral sediments of the Coastal Plain section.

"These lines of structural weakness or fissures were partially sealed by the deposition of the later overlapping strata now capping the oil-pools, but the spasmodic recurrences of the volcanic emanations kept raising and partially replacing the sands and clays of these latter strata to form in spots along these fissures the present mounds and salt islands."

It will be observed that Mr. Hill's "hydrostatic" pressure is here changed to a "volcanic" pressure, and that the saline waters, as well as the hydrocarbon and sulphur gases and vapors, are also referred to volcanic origin, instead of being regarded as derived from meteoric sources and from the strata and sediments beneath the coast prairie.

The statement suggested above, forms a complete geological explanation of the origin of the oil and associated products of the mounds; on the other hand Mr. Hill's explanation is only partial and does not in any way solve the problem of the final origin of the oil, sulphur and salt which he simply admits, without explanation, to be disseminated in more or less minute quantities through the "bituminous sediments of the coastal plain section." Nor does he account at all for the natural gas and the hydrogen sulphide gas, two of the most important associated products of his "oil-phenomena" of the mounds. It cannot, indeed, be admitted that the descending meteoric waters leach or gather also these gases downward through the sediments, to the places of structural weakness or fissures, and then float these gases upward at a few spots along these fissures.

In support of the hypothesis of volcanic origin, the following facts are adduced:

The oils, waters and gases under these mounds are not under hydrostatic pressure. This is amply demonstrated by the fact that the once-famous gushers of Spindle-Top are already gushing no more, and now have to be pumped. If the pressure there were artesian or hydrostatic, they would still be gushing the same as at first; and, if the oil had been exhausted in some of them, these would be gushing water out of the sup-

posed artesian water-column behind the oil. It is now admitted that, in all the oil- and gas-fields, the rock-pressure of the gas is a stored energy continually decreasing as the gas escapes. Gas itself has to be pumped to-day in many fields where its pressure at first was too strong. Surely, this phenomenon is not a result of artesian or hydrostatic water-pressure. Messrs. Hayes and Kennedy say in their recent report on these oil-fields:[8]

"It appears highly probable that the pressure in the oil-reservoir is due largely to the expansive force of the associated gas." But the question remains: To what is the expansive force of the gas due? This cannot be answered except by reference to its real volcanic source. The one word "volcanic" explains it all. And the invocation of this word is here a forced conclusion. We have indeed seen that there is no sign of Mr. Hill's "tremendous hydrostatic pressure"[9] in the wells of Spindle-Top which now have to be pumped, yet Mr. Hill is doubtless right in his conclusion that the oil and associated products are secondary replacements caused by circulating-waters (accompanied by vapors and gases); and, if these waters and gases were not pushed by a hydrostatic pressure, that is, one not from above, then they surely must have been pushed from below by a volcanic pressure.

Mr. Hill[10] says "the association of oil, sulphur, sulphuretted hydrogen gas, gypsum, dolomite and salt, constituting collectively what might be termed the oil-phenomena" of the mounds. Is not that association the unmistakable seal referred to above as belonging to the solfataric volcanic phenomena? And how could that unique stamp be found under these mounds unless they had been percolated by exactly and precisely the same hot waters, vapors and gases which are emitted in great abundance, and at times with disastrous violence and pressure, along the lines of structural weakness or fissures of the volcanic districts of the earth?

It follows that, along lines of faults or of structural weakness of the Texas-Louisiana Gulf Coastal Plain, volcanic gases (aqueous emanations) have issued in spots, like volcanoes along volcanic

[8] Oil-Fields of the Texas-Louisiana Gulf Coastal Plain, *Bulletin U. S. Geological Survey*, No. 212, p. 155; also *Bulletin U. S. Geological Survey*, No. 213, p. 350.

[9] *Trans.*, xxxiii., 398 and 402.

[10] *Trans.*, xxxiii., 396 and 397.

ranges, constituting at each spot an elevating "force acting vertically and lifting a small portion of the earth's crust," as Mr. Hayes[11] puts it, but without explaining what the force was. This force, as Mr. Hayes also recognizes, "was active down to a very recent date," in fact, it may be said to be active now, as in places the waters and oils are still hot and the gases are still issuing. Thus the mounds and salt islands of that region were formed exactly as suffionis, salses and mud-volcanoes form similar deposits to-day in many typical volcanic districts, as, for instance, in Italy[12] and Java,[13] "where clouds of steam also issue from the vents and the heat is intense," and where sulphur, gypsum and salt are also abundantly associated. The only difference is one of intensity, being less pronounced in the Texas-Louisiana region which is further from the center of volcanic activity. An intermediate case between the mounds and salt islands of Texas-Louisiana and the Java mud-volcanoes, is the Trinidad Pitch Lake which the careful researches of Clifford Richardson,[14] have determined to be unquestionably "the crater of an old mud-volcano or geyser" with an influx of soft pitch at the center of the lake, amounting still to thousands of tons yearly.

Throughout his paper, Mr. Hill divides the Texas oil-deposits into two classes: The sheet-oil and the pocket-oil deposits. In the first class he conceives the oil to be indigenous, or nearly so, and to have come from "bituminous" shales in close proximity to the sands. The bitumen (or oil) in the shales, Mr. Hill believes is due to the decomposition of organic matter.[15] But even if that be admitted for the sake of argument, Mr. Hill does not explain how the oil was able to travel out of the impervious shales into the sands. No matter how close the proximity, I think such a movement of the oil out of the shales would be impossible. Of the impervious character of shales, even under great pressure, there is abundant evidence in the oil- and gas-fields, where a few feet or even a few inches of shales prevent all leakages, so that these strata are generally considered as forming, and often do form, the covers

[11] C. W. Hayes, *Bulletin U. S. Geological Survey*, No. 213, p. 347.
[12] *Journal of the Canadian Mining Institute*, vol. vi., p. 86.
[13] *Scientific American*, Oct. 10. 1903.
[14] *The Nature and Origin of Asphalt*, Long Island City, N. Y., 1898.
[15] *Trans.*, xxxiii., 364.

or cap-rocks of the sand-reservoirs. Such covers, sealing hermetically the oil and gas below, cannot logically be regarded as, at the same time, the source from which other sand-reservoirs above have been filled.

In the second class or pocket-oil deposits, the foreign or adventitious nature of the oil and associated products is so glaring, that for the deposits he describes, Mr. Hill gives up the old idea of an indigenous oil made by decomposition of organic matter. But this old notion is retained for all the other deposits of Texas except those of the post-Eocene under the mounds; and even these are made to originate eventually from the same organic source, though in a less direct way, through the intervention of circulating meteoric waters.

The only attempt Mr. Hill makes to prove this indigenous organic origin of the so-called sheet-oil is in these words:[16]

"Of the 22,000 ft. of sedimentaries in the Texas section, all but less than 2,000 ft. are unconsolidated clays and sands, which may be more or less bituminous. Some of the limestones are also bituminous. Oil or bitumen has been found in at least a dozen horizons of the section. . . . In two instances, in Texas, bituminous matter (asphaltum) is found in apparently indigenous beds of lime-shell conglomerate. . . . The oil of the Corsicana field and that of San Antonio is derived from the shales of the Upper Cretaceous. . . . The strata of the Eocene Tertiary present every favorable condition for the generation of petroleum, whether this material be derived from the decomposition of marine organisms, as alleged by some, or from the hydrocarbons generated by vegetable matter, as believed by others. There are vast accumulations of both materials in eastern Texas. . . . Between the Eocene or Claiborne stratum of Nacogdoches and the uppermost Pleistocene stratum of Beaumont, there are thousands of feet of gypsiferous clays and sands in which nature may now be distilling oil."

As will readily be seen, all the above proofs of Mr. Hill are either simply assertions that bitumen or oil exists there (and, of course, that proves nothing as to its origin), or are suppositions against the known facts, "that the oil is derived from the decomposition of marine organisms, or of vegetable matter of which there are vast accumulations in eastern Texas, or is distilled from gypsiferous clays and sands."

Is it not indeed known beyond the shadow of a doubt, that no bodies of marine or any other organisms are, or ever were, entombed in these sedimentaries? Shells or bones of millions of organisms were entombed, but not one body. The same is true

[16] *Trans.*, xxxiii., 400 and 401.

of the sedimentaries forming to-day along thousands of miles of coast-line, from which not one geologist has ever produced one entombed body of marine or any other organism. So much for the "vast accumulations of marine organisms in eastern Texas."

Is it not also an undisputed recognized geological fact that the vegetable matter entombed in these sedimentaries (including the gypsiferous clays and sands) is to be found, and is found to-day, as carbonaceous "undistilled" matter, still containing the bitumens or oils, which it would have lost had the temperature necessary for distillation ever been attained? Either there was distillation of these sediments, or there was not; and the lignites, coals and other carbonaceous matter found abundantly in them to-day are most convincing proofs that there is not, and was not, sufficient heat there to cause distillation, and thus to produce the petroleum.

These simple geological considerations lead to the safe conclusion that the so-called sheet-oil is not an indigenous product of the decomposition of organic matter. That it is also, like the so-called pocket-oil, a secondary product of impregnation and replacement, becomes quite clear when we remember that these so-called sheet-oil deposits are found, not only in a few horizons, but in hundreds, from the oldest Paleozoic to the alluvial gravels and sands of to-day; moreover, nowhere do they spread like sheets, but, on the contrary, they are always found in relatively very small pools in porous reservoir-rocks which are barren of oil outside of the little pools (exactly as in the case of the mounds). These pools form only a very small percentage of the area of the numerous rock-strata in which they are only occasionally found. It is admitted to-day by all who have studied the question, that these accidental pools are only receptacles or reservoirs. That these reservoirs were filled also from the great volcanic tank below, in a manner exactly similar to the case of the Texas mounds, is plain when all the evidence enumerated above which geology brings forward to-day, is considered together.

The above views on the origin of the oil-phenomena, not only of the mounds and salt islands of the Texas-Louisana district, but of all other oil-deposits, suggest a simple interpretation of the geological occurrence of oil which should be a

guide to important results in the practical development of new oil- and gas-fields in the United States, as they have already been such a valuable guide to me in the development of large new gas-fields in Canada."[17] Indeed these views have led me to the following important conclusion:

Oil and gas were only supplied along some of the lines of structural weakness or along some of the fractured zones of the crust of the earth, and, therefore, the new fields are to be found only along these zones or belts.

The numerous oil- and gas-fields known to-day indicate plainly a considerable number of these oil-belts; but more remain to be discovered, and new ones are coming rapidly to the front, especially in the United States. I have long been convinced, on the considerations and for the reasons given above, that this view is the solution of the problem of the geological occurrence of oil and of oil developments and explorations. It follows, therefore, that, as far as practical results are concerned, the important point is to trace accurately these fissured zones or belts on good maps, and to drill in the localities thus indicated.

I have been at work ever since 1888, on maps of this character, embracing North America, and I hope to be able to publish my results before very long, as soon as our present knowledge of these most important structural dislocations is a little more complete.

[17] *Journal of the Canadian Mining Institute*, vol. iii., p. 68.

Geogenesis and Some of its Bearings on Economic Geology.

BY PERSIFOR FRAZER, PHILADELPHIA, PA.

(Atlantic City Meeting, February, 1904.)

"The nebular theory is a noble speculation supported by plausible argument, and the verdict of science on the whole subject cannot be better expressed than in the words of Newcomb :—'At the present time we can only say that the nebular hypothesis is indicated by the general tendencies of the laws of nature, that it has not been proved to be inconsistent with any fact, that it is almost a necessary consequence of the only theory by which we can account for the origin and conservation of the sun's heat, but that it rests on the assumption that this conservation is to be explained by the laws of nature as we now see them in operation. Should any one be skeptical as to the sufficiency of these laws to account for the present state of things, science can furnish no evidence strong enough to overthrow his doubts until the sun shall be found growing smaller by actual measurement, or the nebulæ be actually seen to condense into stars and systems.'" [1]

Leibnitz[2] is credited by Cuvier[3] (who associates with him Descartes) with "amusing their imaginations" by conceiving the world to be an extinguished sun or vitrified globe upon which the vapors, condensing in proportion as it cooled, formed the seas, and afterwards deposited the calcareous strata. Whiston[4] believed the earth was created from the atmosphere of one comet and deluged by the tail of another. The heat which remained from its origin excited the whole antediluvian population, men and animals, to sin, for which they were all drowned in the deluge, except the fish, whose passions were apparently less violent.

De Maillet[5] held that the globe had been covered by water for many thousand years. The terrestrial animals were originally inhabitants of the sea, and man himself began his career as a fish. He adds it is not uncommon to meet with fishes in

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[1] Sir R. S. Ball, LL.D., F.R.S., Astronomer-Royal for Ireland, "Nebular Theory," *Ninth Edition of the Encyclopædia Britannica*, vol. xvii., pp. 311, 312.

[2] *Protogæa. Act. Lips.*, 1683. Göttingen, 1749.

[3] *Discours sur les Revolutions sur la surface du Globe*, Paris, 1725.

[4] *New Theory of the Earth*, London, 1708.

[5] *Telliamed*, 1748.

the ocean which are still half-men, but whose descendants will be perfect human beings.

Buffon thought a violent blow of a comet upon the sun struck off the mass of our earth in a liquefied state, along with the masses of all the other planets of our system, at the same instant. This enabled him to obtain positive dates by calculating how long these fragments had taken to cool, and how far frozen were the smaller bodies.

Burnet[6] thought the whole earth was at first a light crust over the sea which, being broken to produce the deluge, formed mountains by its fragments.

Woodward[7] thought the deluge was occasioned by a momentary supension of cohesion among the particles of mineral bodies.

Kepler thought the globe possessed of "living faculties," instinct, and volition, to the smallest of its particles, attracting or repelling them according to its preferences or antipathies, etc.; and that it also contained a circulating vital fluid.

The nebular theory was founded on the researches of Sir William Herschel, 1784–1818, and La Place, 1796,[8] and was in 1796 first announced as an hypothesis by the latter in the popularized version of the *Mécanique Céleste* referred to in the last note. Herschel thought he had detected in the various nebulæ visible to man a nearly complete series of stages of development from the diffused nebulæ, through those in which a nucleus can just be discerned, to others in which the nucleus is a brilliant star-like point. The transitions from these to the ordinary nebulous stars, and from the latter to the ordinary stars, are natural by a few graduated stages. Yet it must be acknowledged that none of the transmutations required by this theory have ever been seen, as indeed it is not likely they ever could be, since the periods of time required are so incalculably great that the changes effected in a few centuries, or, perhaps, even in millenia, would be entirely inappreciable.

Professor Ball says:[9]

". . . . It is infinitely probable that such a mass should really have *some* motion of rotation. . . . As this vast mass cooled, it must by the laws of heat have con-

[6] *Telluris Theoria Sacra.*

[7] *Essay Towards the Natural History of the Earth*, London, 1702.

[8] *Exposition du Système du Monde*, Paris, 1796.

[9] *Encyclopædia Britannica*, Ninth edition, "Nebular Theory," vol. xvii, p. 310.

tracted toward the center, and as it contracted it must, according to a well-known law of dynamics, rotate more rapidly. . . . The centrifugal force on the outer parts of the mass would more than counterbalance the attraction of the center, and thus we would have the outer parts left as a ring. The inner portion will still continue to contract, the same process will be repeated, and thus a second ring be formed. We have then grounds for believing that the original nebulæ will separate into a series of rings all revolving in the same direction with a central nebulous mass in the interior."

The separation of the rings would most probably occur while the material composing them was still in the state of a gas; otherwise the immense contraction to the size of the central nucleus would be unaccounted for. Up to this point, there is no attempt of the new hypothesis to oppose the old; but at this stage the two commence to diverge. The La Place-Herschel theory supposes these rings to continue to cool and to contract, finally passing from the gaseous to the liquid state. If the consolidation took place with comparative uniformity, a vast number of small planets might be expected, like those between the orbits of Mars and Jupiter. But if the ring were not uniform, some parts would condense more rapidly than others, and the effect would be to draw into a single mass the material of the ring, thus forming a planet which, just as itself had been produced from the central mass or sun, would produce one or more hot gaseous or liquid rings like those of Saturn, or hot gaseous or liquid satellites. Here astronomy left the subject, with a leaning to the supposition that the second method of satellite-forming, namely, the drawing of the entire material of the ring, while still gaseous or liquid, into a new body, was the most usual, if not the only form of evolution. Most of the speculations have proceeded along this line, as, for instance, those of Dana in the chapters of his manual[10] on the "Archean" and on "Dynamic geology."

Even by some modern astronomers who maintain the meteorite composition of Saturn's ring, the separation of the earth and moon has been compared to a fission of the viscous mass like that of the *amœba*, each part withdrawing into an independent globe, as is the case when a drop of mercury is divided.

Dana supposes: a "first era," in which the earth was a globe of molten rock, enveloped in an atmosphere, containing

[10] Dana, J. D., *Manual of Geology*, New York, Ivison, Blakeman, Taylor & Co., 12th ed., 1875, etc.

the dissociated elements of the future waters, and whatever else the heat could throw into a state of vapor; a "second era" in which the cooling proceeded until the earth became solid at the center (Hopkins) through pressure, with a crust outside from cooling, the vapors of the atmosphere were mostly condensed (T. Sterry Hunt), and a nearly or quite universal envelope of water was created; and a "third era" in which the cooling proceeded to 80° or 100° C., admitting the simplest form of vegetable life, etc.

No one ever described more eloquently or with more scientific acumen, Nature's methods in bringing the earth out of the state of the Universe which is supposed to have existed at the epoch when the old nebular hypothesis should end and the new planetesimal hypothesis begins, than our former President, the late T. Sterry Hunt.[11]

He cites Chacornac's hypothesis of an aggregation and concretion proceeding within the primal nebular mass, resulting in the production of sun and planets. He leans to the theory of detached rings, remarking, however, "In either case we come to the conclusion that our earth must have been at one time in an intensely heated gaseous condition and then homogeneous. Solidification commenced at the center." . . . "There is no good reason for supposing that the deeply seated portions have intervened in any direct manner in the production of the superficial crust." All the chlorides, sulphates, etc., would, at the temperature of the first surface-hardening, become silicates resembling furnace-slags and glasses. The atmosphere charged with acid-gases would have been of immense density. The half-cooled crust would be flooded with highly heated solutions of hydrochloric and sulphuric acids. Chlorides and sulphides would form until these were satisfied, and silica would separate out as quartz. The resulting sea-water would hold chlorides and sulphates of calcium, magnesium and sodium, salts of aluminum, and other bases. The atmosphere thus purged of chlorine and sulphur would still have more carbon dioxide than at present.

Next, under subærial decay, the exposed crust would be decomposed by carbonic acid and moisture which would convert

[11] The Chemistry of the Primeval Earth. Friday evening lecture before the Royal Institute of Great Britain, May 31, 1867. *Chemical and Geological Essays*, No. IV., Boston, Osgood, 1875.

the complex silicates into clay, while the carbonates of lime, magnesia and alkalies would be carried into the sea in a state of solution. The first effect would be to precipitate the alumina and heavy metals, after which the calcium chloride of the sea-water would be decomposed producing limestone and common salt.

He adduces the greater content of salts of lime and magnesia, "in the fossil sea-waters still to be found in certain regions imprisoned in the pores of the older stratified rocks," than in modern sea-waters, as a proof of the accuracy of this hypothesis; since "the great deposits of gypsum generally associated with magnesian limestones" furnish proof of the peculiar constitution of the primeval atmosphere, laden with carbon dioxide. Thus, as in so many cases, two opposing schools appeal to the same known facts each in confirmation of its own theory.

Recently, Professor Chamberlin,[12] of the University of Chicago, has propounded a new hypothesis, which has been clearly stated by Professor Herman LeRoy Fairchild,[13] Secretary of the Geological Society of America, and which is called the "planetesimal hypothesis."

According to La Place, our sun had once an immense, hot, nebulous atmosphere, extending as far as, or even farther than, the space occupied by the planets of our system.

The new hypothesis would have the original rings from the contracting solar mass cool and condense with uniformity, and thus produce countless numbers of large and small fragments, down to planetary dust. This development is in conformity with the view of Professor Pickering[14] and other physical astronomers, that Saturn's ring is composed of countless numbers of meteorites of different sizes, each moving in its own orbit. These particles when solidified and cooled would then commence the process of aggregation by mutual attraction, each separate particle either forming a nucleus to which the neighboring particles were attracted, or adding itself to one or the other of such nuclei, until interplanetary space, once full of such "planetesimals," was comparatively cleared of them. This would imply that a few bodies would grow from the

[12] *Journal of Geology*, vi., 609; vii., 545, 667, 751.

[13] Fairchild, *Bulletin of the Geological Society of America*, vol. xv., 243–266.

[14] *Annals of Harvard Observatory*, vols. xxxii. and li.

planetesimal to the planet dimension by accretion from without; and that, by attracting to themselves the outlying smaller bodies, planetary space would be finally nearly depleted of matter. In the meantime, these added bodies, as they increased the mass, would correspondingly increase the pressure and temperature of the growing planet until the more volatile constituents of its interior would be driven up through its substance, thus forming an atmosphere which, however, owing to the too feeble attraction of gravitation, could not be retained until the asteroid had reached or passed the dimensions of our moon. When its attractive force became sufficient, the heavier of the vapors would be retained as an atmosphere, and, subsequently, lighter and still lighter gases would be similarly held; and when the size of the earth was reached, the conditions of increased pressure would have developed heat, perhaps to the point of fusion, at different points within the planet. This heat and pressure would keep continually forcing the more easily vaporizable constituents to the surface, from the epoch when it attained the last-mentioned stage up to the present time; and, if the mass of the planet were sufficient to retain them and prevent their expulsion into surrounding space, the accumulation of these constituents would produce an atmosphere which must be continually increasing so long as heat and vaporizable matter exist in the planet's interior. It would follow from this that the cold cinders, devoid of atmosphere, like our moon, do not represent the end of a cycle of changes during which the astral body has passed successively through conditions of fusion, of temperate heat with an atmosphere; and finally lost its heat by radiation, and its atmosphere by absorption into its porous interior; but a state of arrested development, the nucleal body neither having been picked up by some larger body, nor itself having picked up enough matter from the floating particles in space to reach the mass-limit at which it might have retained an atmosphere.

But inasmuch as all the discrete matter, although sparsely distributed throughout space, is not yet exhausted, there is still hope that a few of these undersized waifs may, after a very long time, take to themselves enough floating substance to produce the pressure and consequent heat necessary to provide themselves with water, metalliferous veins, and an atmospheric

envelope. (This corollary, however, is added by me and not by Professor Fairchild.)

This new hypothesis requires the assumption that "the substances of atmosphere and ocean were originally part of the planetesimals;" but Professor Fairchild thinks it may not be important at present to determine whether they existed as superficial condensations, like occluded gases, or as part of the essential substance; that is, in chemical combinations as hydrates, carbonates, ice, ammonia, nitrates, etc. Throughout geologic time the loss and restoration of carbon dioxide seem to have been well balanced. At first, when the mass of our planet was too small to hold the lighter gases, it is supposed to have been the chief constituent of the atmosphere, but with increase in mass the lighter gases took its place and diminished its volume.

The ocean is constantly being reinforced by water thrown into the atmosphere by volcanic discharges; but it is not necessary to suppose that these supplies exist as water in the hot hypogeal depths. Possibly hydrogen and oxygen, forming other compounds dissociated by heat, may combine together in the volcanic vents. This would, at least, partly explain the dynamic effects of eruptions. The salts held in solution in the ocean are mostly derived, like the water itself, from the earth's deeps, and not from the solution of the decayed and soluble rocks by feeding-streams.

Professor Fairchild's objection to the molten interior, supposed to be a necessary consequence of the nebular hypothesis, that "it is difficult to see why an outflow of molten rock once begun should ever cease," does not seem insuperable; but he makes a good point in asking how the water of the surface could make its way to the interior in opposition to the forces which are continually throwing it out.

The main interest, here, of this new speculation concerns the production of hydro-carbon compounds and others which have been heretofore regarded as of organic origin. One may easily comprehend the vaporization of volatile simple substances and their re-condensation and congelation, but the production of petroleum, albertite, ozocerite, gilsonite, etc., by the dry or furnace process is a startling novelty. The argument to support this view is that if all the carbon represented by the limestones

and by hydro-carbons existed at an earlier period in the earth's atmosphere, there must have been from 20,000 to 200,000 times as much of it there than now. The greater amount of the storage of carbon dioxide in the rocks took place in post-Cambrian, and much in Mesozoic, time. But the existence of air-breathing animals in those periods precludes the assumption that so much carbon dioxide existed in the atmosphere, because it would thereby have been unfitted for breathing. Therefore this material must have been derived from the earth's interior; and the great bodies of bituminous matter, like the petroleum of Baku, and the asphaltum and pitch of Trinidad, must be of immediate derivation from subterranean sources. In proof of this contention are cited—graphite in gneiss and eruptives; liquid carbon dioxide in crystalline rocks; hydrogen and its compounds in volcanic discharges; mofettes, petroleum, etc., in volcanic regions; hydro-carbons in ore-bodies; gas and oil in all horizons, some of them otherwise barren; and the absence of any certainly decarbonated rocks.

The heated water and steam necessary for the production of many ore-veins and deposits must have been derived from below, because meteoric waters could not descend against forces which are continually ejecting them.

Salt and gypsum must be indigenous to the earth's mass, because at Stassfurt and other places, the deposits are so thick and so pure that they could not be the result of evaporation.

It is not my intention to pursue the consideration of this extremely interesting subject further into the questions of climate, glaciation, diastrophy, asymmetry, and biology, though all of these subjects are of great importance; but to say a word as to the relation of this hypothesis to some of those hypotheses which deal with the products of exploitation useful to man.

For one thing, if the new hypothesis prove well founded, there is a greater likelihood of discovering startlingly new things at any time. If the globe be an agglomerated scrap-heap of cold cosmic asteroids, boulders and dust; and man is the Mr. Boffin who makes his living by finding hidden treasures in it, any one may discover a fragment of some unique body (perhaps remelted, perhaps not), which came from some remote region of space where the rarer elements, or elements different altogether from those familiar to us, were formed. It

is true that much of the nebular ring which was rounded into our globe may be hidden in earth's remote depths whither man can never penetrate; but with the volcanic activity of the past we are at least more likely to have an approximate idea of our building-materials if these were a single melt, as in the usually accepted corollary to the nebular theory, than if they represented myriads of melts, as in the new hypothesis. The new hypothesis contemplates the possibility of the filling-up of the ocean basins by cosmic litter more than once, and it is not unsupposable that somewhere a sample ingot or slag might be found which had never been made over by heat, and which should differ from anything yet known.

A recent series of articles by Mr. Eugene Coste of Toronto adds additional interest to this branch of the subject; especially a paper,[15] presented after the joint session of this Institute and of the Franklin Institute at Philadelphia, May 15, 1902, as part of the discussion.

Mr. Coste castigates the views of "many able geologists" as severely as does Professor Fairchild, the blind adherents of a rash guess which never explained anything. The former boldly takes the ground that oil (petroleum) is "inorganic and the result of solfataric volcanic emanations." His reasons are: That animal organic remains are never entombed in rock-formations, and therefore cannot produce oil by distillation there; that vegetable organic remains decompose into carbonaceous matter, peat, lignite, marsh-gas, etc.; that distillation of carbonaceous matter has rarely taken place in nature; and that in volcanic regions gaseous, liquid, and solid hydro-carbons, carbon dioxide, salt, and other chlorides, hydrogen sulphide, sulphur, gypsum, and hot calcic and siliceous waters are found. As to petroleum: In all gas- or oil-fields the gaseous products are under strong pressure which is not artesian or hydrostatic, and which increases in depth, and is therefore volcanic. Heated oils and waters are met with in some fields. The oil- and gas-fields are among faulted and fissured zones. Oil, gas and bitumen are never indigenous to the strata in which they are found. Oil and gas are often found overlying strata so resistant that nothing but volcanic force could penetrate them.

[15] See page 288.

If the point be conceded that oils are the result of volcanic and solfataric action, it will be much easier to reconcile this to an original discrete origin of the globe. As to the ore-deposits, either theory of the origin of the earth will suffice to account for them provided hypogeal water can be supplied.

One advantage of the new hypothesis, according to Professor Fairchild, is that it will release geologists from the oppressive tyranny of physicists and mathematicians. For the latter class especially, "scientific men in general seem to have an almost superstitious regard." Geologists have been obliged, by these exacting meddlers, to cramp themselves into the inadequate trifle of from 350,000,000 to 1,300,000,000 years, because, forsooth, no longer time could be allowed for the geologic history of the earth, by physicists like Helmholtz, etc. Geologists have been too generous and deferential, and they propose from now on to have a real good sufficient time.

While according to this new hypothesis every respect, it is neither necessary to discard with ignominy the nebular theory, nor even the hot fluid earth speculations hung upon it, nor to brand the supporters of these views with the charge of stupidity. It is still difficult to conceive the manner in which the volatile and inflammable hydro-carbons have been deposited, even if their association with vulcanism be established. If it should finally appear that the earth had a "planetesimal" origin; that would not prove that some other physical units in space may not have been derived from the hot fluid state. Indeed the planitesimal hypothesis assumes the cold discrete particles themselves to have been thus derived, and probably neither Professor Chamberlin nor Professor Fairchild would be willing to assign the maximum size of a planitesimal unit.

The continuous supply of water from the extreme depths which have many ages ago attained their maximum temperature is also difficult to understand. All that need be said is that it is a very ingenious hypothesis like the other; but just as recent discoveries have shown both Newton's emanation theory and Huyghens' undulatory theory of light, true in certain cases, while neither of them contains the whole truth, so here we may assume that nature has made solid bodies which gravitate in space both by the fire-fluid and the cold-concentration processes.

It should also be kept in mind that the really important part of the nebular hypothesis conception is neither affected by assuming the correctness of the new hypothesis, nor is its probability in any way involved; but it might be well to delimit the application of the nebular hypothesis to the production of a revolving central body and centrifugal rings.

What happened at this stage to produce the solid bodies now floating in space is not vitally important to the establishment of the evolution of matter from chaos (Crooke's fire-mist, not yet matter) to matter.

Mineral Deposits of Santiago, Cuba.

BY HARRISON SOUDER, PHILADELPHIA, PA.

(Atlantic City Meeting, February, 1904.)

In view of the proposed visit of the Institute to Cuba this winter, the following brief sketch of the principal mineral deposits near Santiago de Cuba has been prepared largely from notes taken in February, 1903, during a visit to the mines then in operation in the province of Santiago de Cuba.

Westward from Cape Maysi, the most eastern point of the island, the coast is rocky and uninviting. The usual course of the steamers is well in toward the shore, and the lines of stratification of the rocks can be distinguished clearly, the presence of iron or other ores being indicated by frequent blotches of mineral stain.

One is impressed with the general rugged character of the southeastern coast-line,—an almost unbroken barrier of coral-limestone bluffs, full of caves and hollows, into which the surf rushes with a roar and bursts into tall fountains of spray. Beyond rises abruptly the Sierra Maestra range, whose sharp, peaked crest-line betrays its eruptive or igneous origin. From a height of about 800 ft. near Cape Maysi, this range gradually rises to the westward, attaining its maximum elevation of more than 8,300 ft. in the Pico Turquino, beyond Santiago. A general resemblance to the mountain-ranges of southern New Mexico is noticeable.

At Daiquiri, 20 miles east of Santiago harbor, the extensive

operations of the Spanish-American Iron Co. are visible. At the time of my visit a number of tramp ore-steamers were standing off-shore, the sea being too rough to permit them to load ore at the company's pier.

The entrance to Santiago harbor is narrow and tortuous, and Morro Castle towers over the steamer on the right. As the city is neared the steamer passes close to the substantial ore-dock of the Juragua Iron Co.

The city of Santiago is one of the most picturesque on the island. At the head of the bay, on a low hill, rising gradually from the water's edge, it commands a beautiful view of the harbor and the Cobre Mountains beyond. The quaintly-built churches and houses, painted various shades of yellow, green or blue (the last color predominating), the red-tiled roofs, the iron-barred, unglazed windows, the narrow streets, clean and well-paved, the ubiquitous "cocheros,"—all are attractive.

There are several good hotels, but the cuisine of the Hotel Venus is most excellent. El Caney, San Juan, and other points of historical interest are easily accessible by carriage. Fig. 1 is an excellent sketch-map of Santiago and vicinity.

The mining-operations in this district are quite extensive, and various deposits of iron-, manganese- and copper-ores are exploited.

Fig. 2 indicates the location of all the principal ore-deposits in the district.

Manganese-Ores.

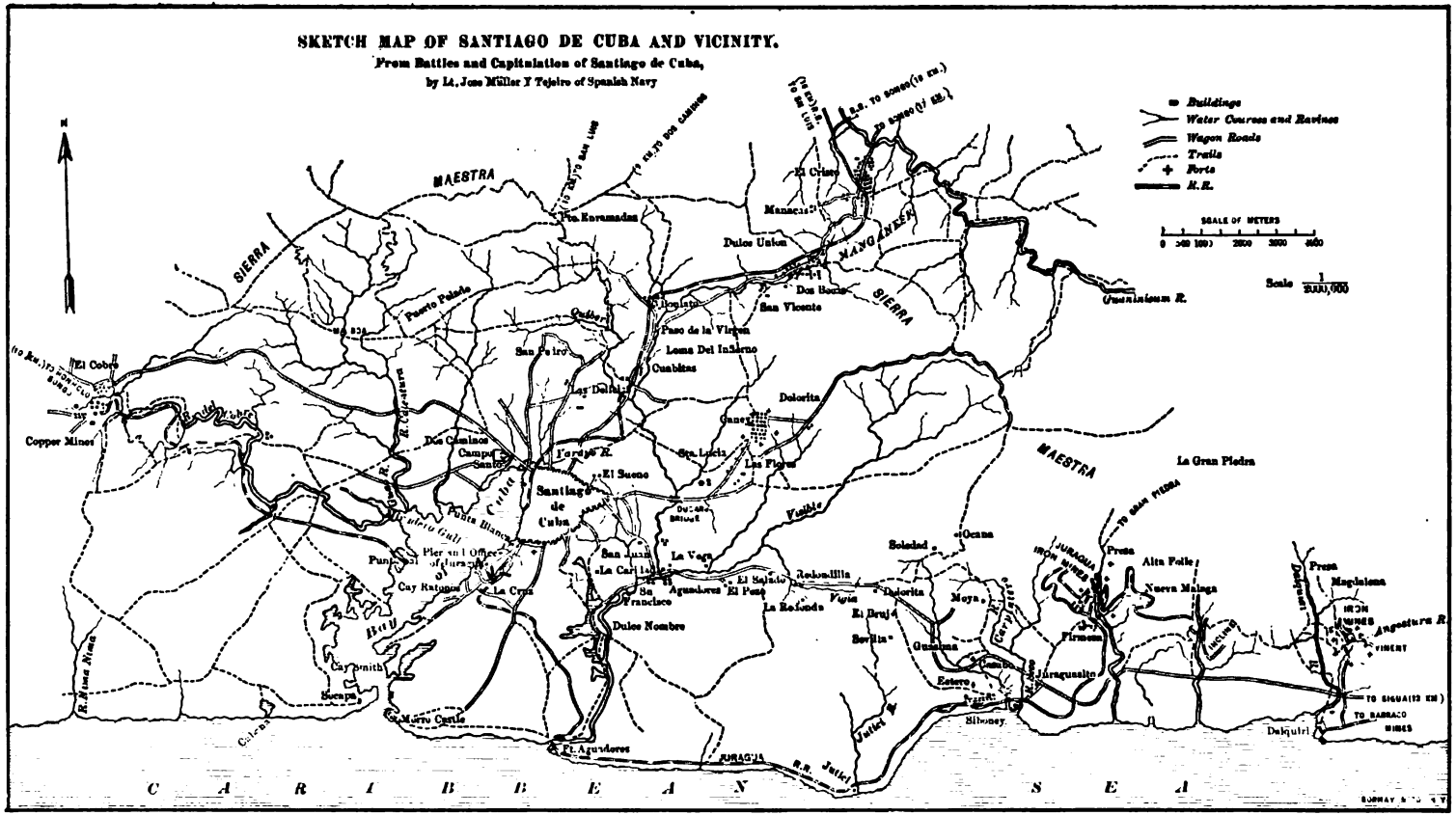
All of the deposits of manganese-ores now being operated are in the San Luis District, and comprise the Ponupo group, the Boston mines, and the Ysabelita group. The last two groups of mines were not being operated at the time of my visit, but it is reported that they have been installed with a fairly complete equipment of railways, washers and concentrators.

The Santiago manganese-deposits are among the most extensive in the Western Hemisphere, ranking about third in the value of output, of which the Ponupo mines furnish by far the greater part.

The Ponupo group is comprised of seven mines, the most important being the Vencedora and Sultana.

The first shipment of manganese-ore was made in 1895, and the quantity has increased gradually, until in 1903 the ship-

FIG. 1.



SKETCH MAP OF SANTIAGO DE CUBA AND VICINITY.

ments amounted to 2,000 tons per month. The ore-bodies are at the surface, requiring but little or no stripping; and while they are of irregular shape, the pockets are large. One cut in the Vencedora mine, 30 ft. wide, with a depth in ore of 30 ft., is shown in Fig. 3. The ore is imbedded in clay and associated with jasper and porphyry.

The ore is chiefly pyrolusite, of which some beautiful crystals are found. Portions of the pockets contain a high percentage of phosphorus, but ore of this quality, however, is easily detected and separated as it is mined.

The Sultana workings shown in Fig. 4, which were being opened up in February, 1903, were most promising. The deposit appears to be a continuous bed of ore extending over several acres, and outcrops may be seen in the steep bluff of

FIG. 2.

SKETCH MAP OF ORE-DEPOSITS NEAR SANTIAGO DE CUBA.

the Ponupo river. Mr. Chibas places the ores of this group in four classes, as follows:

1. Ore containing mostly pyrolusite.
2. Steel gray ore, of good appearance but of light weight.
3. Wash ore, mixed with clay or soil.
4. Ore high in silica.

The ore is collected in cars running on narrow-gauge tracks, and is brought to the washers by a system of inclined hoists. The two log-washers have a capacity of approximately 35 tons each. The washed ore, loaded into cars by hand and shipped to Santiago, averages from 47 to 48 per cent. of metallic manganese. The Ponupo Mining and Transportation Co. operat-

ing these mines pays a royalty of 9 per cent. of the market-value on unwashed ore and 7.5 per cent. on washed ore.

Manganese-ore is found in several other districts in Santiago de Cuba, notably at Los Negros, Dos Bocas and Majaba hill; but reports concerning the character and extent of these deposits are not very favorable, and transportation facilities are lacking.

Table I. is composed of analyses which will give an idea of the character of the Santiago manganese-ores.

TABLE I.—*Analyses of Manganese-Ores from the Santiago District.*

Components.	Boston Mines.(a)		Ponupo Mines. (b)			
			Vencedora Mine.		Sultana Mine.	
	Sample No. 1.	Sample No. 2.	Sample No. 1.	Sample No. 2.	Sample No. 1.	Sample No. 2.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Metallic manganese....	54.025	56.888	49.19	51.91	51.36	44.63
Metallic iron.....	1.640	0.300	2.42	0.79	0.79	1.77
Phosphorus.....	0.059	0.030	0.062	0.049	0.07	0.038
Sulphur.....	.....	0.005	.....	.....	.....	.....
Silica.....	5.400	0.810	2.72	1.09	1.22	6.11
Alumina.....	1.911	.....	.....	.....	.....	.....
Barium oxide.....	1.105	.....	.....	.....	.....	.....
Moisture.....	0.600	2.000	.....	.....	.....	.....

(a) *Commercial Cuba*, by Wm. J. Clark.
(b) *U. S. Geological Survey*, Production of Manganese-Ores.

The latest reliable figures show an export of manganese-ores from the Santiago district for the year 1902 amounting to 39,628 long tons, which brings the grand total for the years 1888 to 1902 inclusive, to 179,275 long tons.

COPPER-ORES.

Copper-ores are found in several parts of the province, but practically all the ore mined has come from the "El Cobre" deposits, now being rehabilitated.

Copper-mining bids fair to become once more a most important item in Santiago's industrial advancement.

The Cobre mines were opened in 1530 and operated by the Crown for 200 years, and then abandoned. An English company opened them in 1830, but abandoned them at the out-

break of the "Ten Years' War," in 1868. The pumping-plant was destroyed and the mines were flooded. It is estimated that more than 600,000 tons of ore, valued at from $50,000,000 to $60,000,000, was taken from these mines between 1830 and 1868.

In 1902 an American company took hold of the property and at once proceeded to install a modern mining-plant, with suitable pumps, hoists, etc. Pumping was started in November, 1902; and in February, 1903, the water had been lowered 240 ft. Recent reports state the water in the shaft to have been lowered 500 feet.

The character of the deposit thus far opened up is very favorable, and several shipments of ore have been made. It is claimed that the shaft is 1,080 ft. deep, and that the underground workings are about 28 miles in extent. As the water is pumped out it is run over a series of "Californias," or shallow wooden precipitating-basins of 20,000 sq. ft. area, containing scrap-iron (Fig. 5), which yields a high-grade cement-copper.

The ore-vein at its outcrop on the surface shows a width of probably 40 ft. (Fig. 6). The ore is largely malachite and black copper oxide; it contains also chalcopyrite. These mines are 12 miles west of Santiago harbor, and are connected with it by a mule-tramway which is to be re-constructed for operation by steam-power.

Native copper is said to have been found at Dos Palmas, but it is in small quantities and much scattered.

Iron-Ores.

The iron-mines now being worked are east of Santiago. Practically all of them are found at the summit of a range of foot-hills lying between the Sierra Maestra and the coast. The ore-belt begins at Sevilla and extends 18 miles eastwardly to the Sigua deposits. The ore occurs as magnetite and as various forms of red hematite.

The first iron-mine in Cuba was denounced in 1861, but operations did not begin until 1883, and the first ore was shipped in 1884 by the Juragua company. The Juragua iron-mines are 18 miles east of Santiago; the Spanish-American group 5 miles east of these; 8 miles further east is the Berraco

group, operated by the same company; and 5 miles beyond are the Sigua mines.

The Sigua mines were opened in 1902, but few shipments of ore were made. I believe the property has recently been acquired by the Spanish-American Iron Co. The ore is reported to be red specular hematite, containing:

	Per Cent.
Metallic iron,	58.40
Phosphorus,	0.02
Silica,	13.36

At Guama, 40 miles west of Santiago, the Cuban Steel Ore Co. began operations on a large scale in 1900 and built extensive docks, standard-gauge railroad, etc., with the intention of working the Bacardì mines. The operations were abandoned shortly after they were commenced.

The Juragua Iron Co. has 24 miles of narrow-gauge railroad, connecting the mines with a steel shipping-dock in Santiago harbor. For a large part of the distance the railroad follows the shore-line at the foot of the coral cliffs, passing through Siboney, of historic interest. The ore-deposits appear to be great lenses embedded in eruptive rocks—syenite and diorite. The ore is mined, or rather quarried, in open cuts, usually in benches 40 or 50 ft. high, though at the East mine, Fig. 7, the face is 200 ft. high. This company owns or leases the Sevilla, Juraguasito and Firmeza groups; and is operating the East mine, the West mine, and in 1901 opened up the Columbia and Colon mines. These mines are all in the last two groups.

The average analysis of about 30,000 tons of ore from these mines is said to be as follows:

	Per Cent.
Iron,	60.510
Silica,	9.786
Sulphur,	0.328
Manganese,	0.255
Alumina,	2.290
Lime,	1.610
Magnesia,	0.820
Moisture,	1.350
Phosphorus,	0.028

The Spanish-American Iron Co. made its first shipment of iron-ore in 1895. It operates the Lola, Magdalena, Provi-

FIG. 3.

PONUPO MINES: SHOWING ORE-BODY IN VENCEDORA MINE. THE SHAFT IS 15 FT. DEEP IN ORE.

FIG. 4.

PONUPO MINES: THE SULTANA MINE.

FIG. 5.

EL COBRE MINE: VIEW OF NEW HOISTING- AND PUMPING-PLANT, WITH "CALIFORNIAS" OR PRECIPITATING-TANKS IN FOREGROUND.

FIG. 6.

EL COBRE MINE. THE CUT SHOWS WIDTH OF VEIN AT SURFACE.

FIG. 7.

EAST MINE OF THE JURAGUA IRON CO. SHOWING METHOD OF WORKING THE ORE-BODY.

FIG. 8.

MINES OF THE JURAGUA IRON COMPANY.

FIG. 9.

GENERAL VIEW FROM VINENT OF MINES OF SPANISH-AMERICAN IRON COMPANY.

FIG. 10.

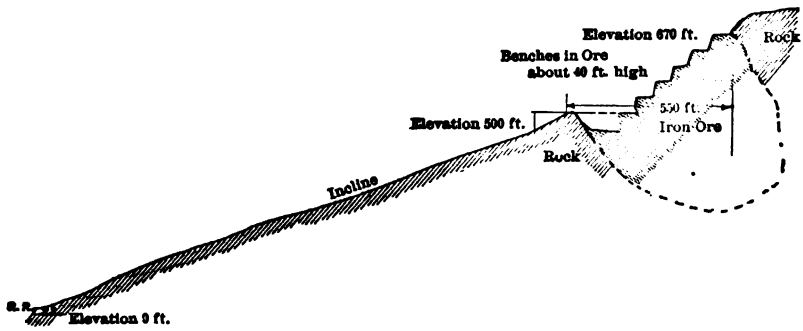
LOLA MINE OF SPANISH-AMERICAN IRON COMPANY.

dencia, San Antonio, Berraco and Fausto groups of mines. A general view of these properties is shown in Fig. 9.

The Berraco and Fausto mines had not been opened up sufficiently last year to determine positively their character and extent, though enough ore was found to warrant the building of 8 miles of railroad of difficult construction. Subsequent developments have proved more than satisfactory. Two analyses of the Berraco iron-ore are, as follows:[1]

	Sample No. 1. Per Cent.	Sample No. 2. Per Cent.
Iron,	65.1	68.05
Silica,	0.065	2.49
Phosphorus,	0.036	0.042

FIG. 11.



THE METHOD OF EXCAVATING THE ORE AT THE MINES.

The other iron-ore deposits, similar in character to those of the Juragua Iron company, have been fully and more systematically developed. Stripping is done with steam-shovels, and the series of benches established gives the opportunity to extract large quantities of ore at a low cost.

At the San Antonio and Lola (Fig. 10) mines, at least six levels are operated (Fig. 11). About one ton of rock is handled for every ton of ore extracted. The ore is carried from the faces in side-dump cars hauled by mules, and is dumped into a series of chutes leading to the lowest level where storage-bins are provided. From these bins the ore is emptied into skips and lowered on inclined planes to the bins, at the foot of the mountain, where the ore is discharged into 10-ton drop-

[1] *Commercial Cuba.* By William J. Clark.

bottom cars in which it is transported to the shipping-pier at Daiquiri, 5 miles away.

The pier is constructed of steel carried on iron cylinders filled with concrete. Bins of about 3,000 tons capacity are provided. The ore is shipped in tramp steamers carrying from 3,000 to 5,000 tons; but they cannot be loaded in rough weather.

It was at this pier that the United States troops landed during the recent Spanish-American war.

At Daiquiri the Spanish-American Iron Company has extensive machine-shops and storage-bins, and it is now constructing a large storage-yard, by erecting several trestles and bridge-spans, where 40,000 tons can be stored. Practically all the labor in these mines is imported from Spain; and the wages are equivalent to $1 (U. S. currency) per day.

An interesting piece of railroad-engineering is a narrow-gauge road to the Berraco mines. It runs through a very difficult country, covered with an almost impenetrable growth. The bridge crossing the Berraco river is the highest in Cuba. It is a steel lattice-truss of 100 ft. span, 125 ft. from river to rail. At the terminus of this road an inclined plane runs up the mountain to a high, level railroad 2 miles long running to the Berraco and Fausto mines.

More than 2,000,000 tons of ore have been taken from the mines of the Spanish-American Iron Co., and there is possibly as much more in sight. The ore is hematite, averaging for the whole group for the year 1902:

	Per Cent.
Iron,	62.25
Silica,	8.05
Phosphorus,	0.032
Sulphur,	0.08
Copper,	0.30

The copper is associated with the ores from the Magdalena mine.

At Camoroncite, 56 miles from Santiago, there is said to be a deposit of iron-ore containing 68 per cent. of magnetic iron.

The following interesting figures of the iron-ore shipped from the Santiago mines are taken from the report of the U. S. Geological Survey.[2] The quantities are in long tons.

Year.	Juragua.	Sigua.	Spanish American.	Cuban Steel Ore.	Total.
1902	221,039	———	455,105	23,590	699,734

[2] *Mineral Resources* for 1902. U. S. Geological Survey, p. 73.

The total quantity of ore shipped from this district is as follows:[3]

	Long Tons.
Juragua Iron Co., 1884 to 1902 (inclusive), . . .	3,911,795
Sigua Iron Co., 1892 to 1893 (inclusive), . . .	20,438
Spanish-American Iron Co., 1895 to 1902 (inclusive), .	1,777,118
Cuban Steel Ore Co., 1901 to 1902 (inclusive), . .	41,241
Grand total, ...	5,750,592

The following is a list of the mining concessions issued by the Provincial Government of Cuba for Santiago Province:[4] Iron, 167; manganese, 141; copper, 81; gold, 9; lead, 9; coal, 7; asphalt, 8; antimony, 1: mercury, 2; zinc, 3; blende, 4; petroleum, 3; total, 435. Many of these are simply concessions on which no work has been done, and the list is given to show the activity of the mining interests in this Province.

References are appended for the benefit of those who are more particularly interested in the mineral industry of Santiago de Cuba. Some of these publications have been consulted in the preparation of this paper:

The Iron-Ore Range of the Santiago District of Cuba, by Prof. Kimball. *Trans.* xiii, 613.

Mineral Resources of Cuba. *Civil Report of the Military Governor of Cuba*, vol. v., Jan. 1 to May 20, 1902.

Production of Manganese-Ores. *Mineral Resources.* U. S. Geological Survey.

Production of Iron-Ores. *Mineral Resources.* U. S. Geological Survey.

Opportunities in Cuba, by Hon. Perfecto Lacoste.

Commercial Cuba, by Mr. William J. Clark.

[3] *Mineral Resources for* 1902, U. S. Geological Survey, p. 73.

[4] *Opportunities in Cuba*, by Hon. Perfecto Lacoste.

Additional Remarks on Surveying-Instruments.

BY H. D. HOSKOLD, BUENOS AIRES, S. A.

(Atlantic City Meeting, February, 1904.)

Thornton's Improved Miner's Dial.

This instrument, shown in Fig. 1, is not generally known, though it has been used in England, and found very practical, handy and useful in work not requiring a transit or theodolite. It is not only a good miner's dial, but a circumferentor as well; that is, it can measure horizontal angles like a theodolite. The magnetic bearings are conveniently read on the inward-beveled upper divided circle; while another circle, graduated on the base-plate, is used for reading the horizontal angles with a vernier. Two leveling spirit-bubbles are sunk in the same base-plate. The vertical divided semicircle is attached to the compact-box by two short axes, which permit the semicircle, with its plain sights, to be turned down to a horizontal position, and rest upon a pin, which relieves the axes from strain. This adjustment gives a clear view of the needle and vernier, and protects the instrument from injury, when carried through low mine-workings.

A groove in the semicircle receives a sliding-bridge which carries the plain sights, and a vernier for reading vertical angles. A ball-and-socket-headed tripod is used; but sometimes the instrument is mounted on parallel-plates and leveling-screws; and sometimes a telescope is attached, instead of the plain sights.

The instrument is admirable for all such Mexican surveys as Mr. E. A. H. Tays describes;[1] and saves so much time and labor as to be vastly superior to the less accurate hanging-compass called by him a "bully" instrument. If the hanging-compass was ever used in England, it has long since given way to more modern and superior instruments.

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[1] *Trans.*, xxxiii., 1035.

The Grubb-Davis Mine-Surveying Dial.

This recently introduced instrument is shown in Fig. 2. It has an exterior horizontal divided circle with verniers, like a theodolite, reading to either three minutes or one. The vertical semicircle, graduated up to 90°, is read with a vernier upon an upward-projecting arm, with clamp and tangent-screw motion; and with a new form of sight, invented by Mr. Grubb, which is said to be a great improvement upon the

Fig. 1.

Thornton's Surveying Dial-Circumferentor.

usual form. The semicircle is screwed to the outside of the compass-box, and magnetic bearings are taken in the usual way. The instrument is mounted with a modified form of Hoffman's patent joint and with leveling-screws; and is sometimes constructed without a circular compass, becoming then a very superior circumferentor, with a long trough magnetic compass below the horizontal circle, for determining the magnetic meridian.

MICROMETRIC STADIA-MEASUREMENT.

Mr. Brough, in his "further discussion,"[2] argues that neither Gascoigne (1639), Huygens (1659), Malvasia (1662), nor Auzout and Picard (1667), "contemplated the direct measurement of distances on the earth by means of the visual angle," with the micrometer; an idea, however, clearly contemplated, in his

FIG. 2.

GRUBB-DAVIS MINE-SURVEYING DIAL AND CIRCUMFERENTOR.

opinion, by Montanari in 1674, as well as later by James Watt in 1771, and William Green in 1778. But Townley, in 1667, distinctly says[3] that Gascoigne "had, before our late Civil Wars, . . . for some Years made use of it [his instrument] not only for taking the Diameter of the Planets and Distances upon

[2] *Trans.*, xxxiii., 1037.

[3] *Trans.*, xxxi., 26, and *Phil. Trans.*, No. 25, p. 457 (May, 1667).

Land, but," etc. This evidence seems to establish the fact that Gascoigne was the first English inventor and user of a micrometer eye-piece, and consequently of a tacheometric and stadiametric principle, for finding distances upon land. The only pertinent and effective contradiction of it would be the demonstration that Townley was mistaken in his statement. Mr. Brough alludes to the death of Gascoigne at Marston Moor (1644), and says that he was then 23 years old. In 1639, therefore (when Townley dates his invention), he was 18 years old. Did Townley make his statement in 1667, twenty-three years after Gascoigne's death, and twenty-eight years after the said invention, as an inference from Gascoigne's published correspondence; or upon his own personal knowledge, or other direct evidence, of Gascoigne's actual use of his micrometer? This question is covered by the references given by me in a former contribution,[4] in which authorities were cited, fully confirming Townley's statement. I may here say that Townley's statement, made at a meeting of the Royal Society in July, 1667, was accompanied with the exhibition of Gascoigne's own micrometer, which he had personally shown in 1639 to Crabtree. Concerning this instrument, Crabtree wrote to his friend Horrocks:

"The first thing Mr. Gascoigne showed me was a large telescope amplified and adorned with inventions of his own, whereby he can take the diameter of the sun and moon or any small angle in the heavens or upon earth most exactly through the glass to a second."

In my former contribution, just cited, I gave a statement on this question, which stands, so far, unimpugned. I may safely conclude that my attribution of this invention to Gascoigne is established beyond question.

On the other hand, it is true that Gascoigne's micrometer eye-piece contained no spider-lines (though its construction permitted the close measurement of angles without them); and it may be that he never applied it to a practical and portable surveying-instrument, for measuring distances on land. Perhaps such measurements, if made by means of it (as Townley's words certainly intimate), were made from a stationary astronomical observatory, to the telescope of which the micrometer

[4] *Trans.*, xxxi., 26.

was attached. Doubtless, if Gascoigne had not died so young, he would have developed practically the use, in land-surveying, of the invention he had made; and, if he had had time to do so, it is not likely that the practical, portable surveying-instrument, with a distance-measuring micrometer, which he would have then introduced, would have fallen into disuse and oblivion. It may be presumed, therefore, that Gascoigne's early death prevented the development of this essential stage of improvement in practice; and this consideration, while, on the one hand, it warrants me in maintaining the claims of a young genius thus cut off in mid-career by a tragic fate, disposes me, on the other hand, to recognize most heartily the merits of such later inventors as Green and Watt, who revived, perfected and applied, after more than a century, the discoveries of their young predecessor.

These questions of "priority" in invention ought not, in my judgment, to be discussed, as if the credit given to some early pioneer were thereby necessarily taken from later and more successful ones. The latter can well afford to recognize the initiative taken by the former; and it is our duty to recognize the merits of both.

Sulphide-Smelting at the National Smelter of the Horseshoe Mining Co., Rapid City, S. D.

BY CHARLES H. FULTON AND THEODOR KNUTZEN, RAPID CITY, S. D.

(Atlantic City Meeting, February, 1904.)

THE plant of the National Smelting Co., a corporation controlled by the Horseshoe Mining Co., was built during 1901 to smelt the dry siliceous ores of the northern Black Hills, extracting the gold- and silver-values in a matte of low copper-percentage which is shipped to Omaha and Denver for refining. Originally, the plant was designed to collect the values of the siliceous ores in an iron-matte, which was to be resmelted with lead-ores, in a lead-furnace, the lead-bullion produced to be refined into Doré bullion in English cupelling-furnaces. However, the scarcity of lead-silver ores in the Black Hills, owing to the present non-productiveness of the Galena district, led to

the abandonment of this plan; and both shaft-furnaces of the plant were run as matting-furnaces, the mattes produced being shipped as above stated.

It is not our intention to present anything very new, but rather to supplement the interesting and valuable paper, Pyritic Smelting in the Black Hills, by Dr. Franklin R. Carpenter.[1]

Smelting in the Black Hills is a difficult matter from a commercial point of view, owing to the fact that the only productive material going into the furnace, generally, is the siliceous ore and a little copper-ore; the pyrite, or pyrrhotite, and the limestone, being barren of values, and no lime-ores, or gold- or silver-bearing pyrite being at present available in the Black Hills. The recent price of coke from the East or from Colorado has also been prohibitive ($9.50 per ton), and Cambria, Wyoming, coke ($4.50 per ton) is of such inferior quality that it cannot be used alone, but has to be mixed in the proportion of 2 to 1 with Eastern or Colorado coke, to be able to smelt with it at all.

The National Smelting plant is situated at the eastern end of Rapid City, on a terrace-site on a spur of the Fremont, Elkhorn & Missouri Valley Railroad, a branch of the Chicago & Northwestern Railway system. Directly below the railroad-trestle are nine 125-ton bins: 3 for siliceous ore, 2 for limestone, 2 for pyrite, 2 for coke, and 1 for coal. The bottom of the bin slopes 50°, the planking being protected from wear by railroad-iron, placed transversely every foot.

The material is shipped to the smelter in 20-ton ore-cars, usually having a bottom-discharge. It is sampled by shoveling, on a sampling-floor at the top of the bins. Lots of 60 tons or less are sampled by taking every fifteenth shovel, while those in excess of 60 tons are sampled by taking every twentieth shovel. The sample is thrown down a chute at the side of each bin, carried by a barrow to the sampling-works, and is there crushed in a 9- by 15-in. Blake crusher which divides it into halves by an "A" discharge, one-half going directly to a pair of 24- by 12-in. Allis-Chalmers rolls. The discharge from these rolls is re-shoveled, every fifth shovel being taken as the

[1] *Trans.*, xxx., 764.

sample, or every tenth shovel, if the ore lot is more than 60 tons. The sample obtained in this way is crushed in a pair of 12- by 12-in. sample-rolls, then coned and quartered on a plate-floor, and the resultant sample ground in a sample-grinder. The moisture is determined at once on the sample-floor, in a specially provided drying-cupboard.

Furnaces.—There are two blast-furnaces; one, a copper-matting furnace 144 by 48 in. in cross-section at the tuyeres, with a 12-in. bosh. The height from the hearth to the feed-floor is 15 ft. The distance from the tuyeres to the bottom of the rectangular gas take-off is 9.5 ft. This take-off is at one end of the furnace, and is 48 by 36 in. in size, the top being 16 in. below the throat-plate.

The other furnace was originally a lead-furnace 120 by 36 in. in cross-section at the tuyeres, with a 12-in. bosh, and 15 ft. high from the hearth to the feed-floor. The well of this furnace has been bricked up, and the furnace run as a copper-matting furnace. The other dimensions are the same as those of the larger furnace, except that the take-off is 44 by 34 in. in size. Both furnaces have 5-in. tuyeres. The hearths of the furnaces are placed on trucks. The furnaces are semi-mechanically charged by means of a specially designed bottom-dump charge-car which runs directly over the throat of the furnace. This method of charging has been found eminently satisfactory, and no trouble from fume is experienced on the charge-floor. The furnaces, as they stand, are somewhat too low, and might, with profit, be increased about 4 ft. in height and still not impair seriously their high concentration.

Steam is furnished by two 200-h.p. Stirling water-tube boilers, with a steam-pressure of 125 lb. per square inch. A No. 8 Green blower furnishes blast for the large furnace, and a No. 7 blower of the same type for the small furnace. Both blowers are directly connected to horizontal engines on the same bed-plate. The blowers run from 130 to 150 rev. per min., furnishing blast at a pressure of from 14 to 18 oz. per square inch. The blast main from the large blower is 30 in. in diameter, and that from the small one 24 inches.

The coal is of a very poor grade, being slack from Cambria, Wyo., which costs $2.75 per ton, delivered at the plant. The ashes are sluiced from the boiler-plant through launders to the

slag-dump. The water-supply of the plant is obtained by pumping from Rapid creek, a few hundred feet below the plant.

Slags.—Table I. gives the analyses of typical slags made at the plant.

TABLE I.—*Analyses of Slags.*

Kind.	Silica.	Ferrous Oxide.	Lime.	Alumina.	Zinc Oxide.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Blowing-in slag.....	42.65	16.67	30.7	6.85	.....
Typical slag.....	47.5	18.7	28.25	3.5	.....
Typical slag.....	50.2	16.35	28.30	4.2	.....
High alumina and zinc.....	42.86	17.24	26.61	9.39	2.28

The precious-metal contents of the slags given in Table I. are usually: Gold 0.01 oz., and silver 0.20 oz. per ton. A trace of copper is also present.

The limestone used as a flux is very pure, and contains only a trace of magnesia, as is shown by the analyses given in Table II. Some magnesia, up to 8 or 10 per cent. replacing lime, is desirable, owing to its greater silica-saturating power, and the lesser specific gravity of the resultant slag. The Golden Reward plant at Deadwood uses a magnesian limestone successfully. At the present time no limestone containing magnesia is available in the vicinity of Rapid City.

The slags are fluid and flow readily from the furnace with a slight arch. They chill quickly, indicating a rather high temperature of formation. Water-cooled trap slag-spouts have been tried on the furnaces, but had to be discarded owing to their marked chilling-effect on the slag. The greater part of the slag, granulated by waste-water from the furnace-jackets, is discharged to flat-cars of the railway which utilizes it as road-ballast. The slag flows from the settling-pot in a thin stream, falling from a height of 4 ft., and strikes the water which flows in a heavy cast-iron gutter of semi-ellipsoid section, 8 in. wide and 6 in. deep, inclined 3 in. per foot for the first 10 ft. The section of the gutter beyond the first 10 ft. is of a somewhat larger cross-section, and inclined but 1 in. per foot. Owing to excessive wear, the section of the gutter where the slag strikes it has to be frequently renewed.

Heated blast is used in smelting. The blast-heating apparatus is a U-pipe stove, containing 12 U-pipes each 16 in. in

diameter and 10 ft. high. This type of stove is not as efficient as it might be, owing to the difficulty in preventing leakage. The stove is placed in the dust-chamber directly beneath the downtake, it being intended to heat the blast only by the waste heat from the furnaces. Under ordinary conditions the temperature of the blast, taken at the tuyeres, is 131° F., with the outside air at 65° F. With the cupelling-furnaces running on some experimental work, the temperature of the blast at the tuyeres was as high as 320° F., the outside air being at a temperature of 77° F.

The flue of the plant is of the zig-zag type, 250 ft. long, extending up hill to a plate-iron stack, 10 ft. in diameter at the bottom, 9 ft. in diameter at the top, and 166 ft. in height from the bottom of the flue where it merges into the stack. The total height from the tuyere-level to the top of the stack is 275 feet.

TABLE II.—*Composition of Materials of Furnace-Charge.*

Name of Material.	Silica.	Alumina.	Ferrous Oxide.	Iron.	Lime.	Magnesia.	Sulphur.	Carbon.	Copper.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Montezuma pyrite	19.35	7.63	.....	31.7	2.57	0.30	32.3	6.50	.....
Bion pyrite	23.95	.....	.....	31.75	.....	.....	37.03	.....	.....
Penobscot ore	61.44	29.2 (a)	.....	.....	5 9	trace	3.01	.....	.....
Ben Hur ore	62.56	14.7 (a)	.....	.....	14.54	.....	.....	.....	.....
Ben Hur ore	73.11	12.36	7.18 (b)	.....	7.2	.....	.....	.....	.....
Limestone	1.94	0.68 (a)	.....	.....	53.61	0.68	.....	.....	.....
Montana copper-ore	31.62	.. ..	.....	21.47	.....	.....	22.85	.....	24.16
Ashes:									
Cambria coke, 80 per cent	36.57	40.35	14.1	.....	4.01	.....	.....	.....	.....
Fairmount coke, 12.5 per cent.	60.46	.....	37.09	.....	2.75	trace	.....	.....	.....

(a) Includes ferric oxide. (b) Ferric oxide.

The Penobscot ore contains from 0.88 to 0.96 oz. of gold and from 1 to 2 oz. of silver per ton. The Ben Hur ore contains from 0.73 to 0.80 oz. of gold and 1.5 oz. of silver per ton. Generally the ore-value is kept less than $20 per ton.

Matte.—The first matte made is low in copper, containing from 10 to 14 per cent. of copper and from 4 to 5 oz. of gold and from 6 to 7 oz. of silver per ton. The matte-fall figured on the total furnace-charge is from 4 to 5 per cent., this fall being amply sufficient to collect the values. We believe a 3 per cent. matte-fall would be sufficient. It was found by experience that some copper was absolutely essential in order to have the matte

collect all of the values and, at the same time, to produce sufficiently clean slags. In 1901, shortly after the plant was started, it was endeavored to run without copper-ores, owing to the difficulty of procuring them. Mattes were made with only a trace of copper in them, but the slags invariably contained from $1 to $2.50 in value of gold per ton. This loss being too great for profit, copper-ores had to be procured. Upon the addition of copper-ores to the furnace, this abnormal loss of value in the slag disappeared, it dropping to the normal value of from 20 to 30 c. per ton and often less. It has been demonstrated by Dr. R. Pearce[2] and E. G. Spillsbury[3] that iron sulphide will not collect gold and silver. Metallic iron will collect gold, but practically no silver, as Dr. F. R. Carpenter has pointed out[4]—a fact which is amply proven by the sows, or metallic-iron accretions, formed in the hearth of the furnace as well as in the fore-hearth. The matte formed rarely contains more than 30 per cent. of sulphur, while the iron monosulphide contains 36.36 per cent., so that the matte is evidently a subsulphide. It also contains metallic iron, which can readily be abstracted by the magnet. We agree with Dr. Carpenter, that it is this metallic iron in the matte which collects the gold, but, unfortunately, it is rarely present in the matte in sufficient quantity to give clean slags. Paradoxical as it may seem, the quantity of metallic iron formed in the furnace is due largely to the amount of oxidation which takes place in the furnace. This point is referred to under the section devoted to sows, on page 335.

The amount of oxidation being difficult to control, the matte-composition and matte-fall vary from time to time. There are occasions when practically no matte is being made, but at the same time, the slags do not increase in value, showing that while no matte is made, metallic iron is being produced. This condition of affairs occurs during periods of much oxidation, and is usually remedied by the charging of extra quantities of pyrite, in order to furnish more sulphur and leave some to remain in the matte.

Accretions on the furnace-walls will decrease the quantity of matte made, by raising the zones of oxidation. Of course, the

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[2] *Trans.*, xviii., 454. [3] *Trans.*, xv., 767. [4] *Loc. cit.*

quantity and pressure of the blast, also, greatly influence the matte production.

Desulphurization.—The matte-fall being generally but 4 or 5 per cent. shows the great desulphurizing action of the furnace, which amounts ordinarily to from 70 to 77 per cent. Taking the typical workings of a day, the quantity of sulphur fed into the furnace in the shape of pyrite was 7,350 lb. and the sulphur in the matte was 2,100 lb., showing a loss of sulphur of 5,250 lb., which is equivalent to 71.5 per cent. of the total quantity of sulphur in the materials charged into the furnace.

The desulphurization-figures for December, 1903, were as follows:—The quantity of sulphur fed into the furnace in pyrite, 175 tons; in matte, 37.5 tons; in copper-ore and concentrates, 27 tons, a total of 239.5 tons. The sulphur in the matte produced was 52 tons, showing a loss of sulphur of 187.5 tons, or 78 per cent. of the quantity charged into the furnace.

The sulphur in the materials carried over mechanically in the flue-dust may be disregarded on account of its relatively small quantity and the fact that much of it is in an oxidized condition. The total quantity of flue-dust produced in treating 3,952 tons of charge amounted to 395.2 tons, or 10 per cent., having an average sulphur-content of 3 per cent., which is equivalent to 11.8 tons of sulphur, a large portion being in an oxidized form.

TABLE III.—*Composition of Mattes.*

Kind.	Iron.	Copper.	Sulphur.	Zinc.	Gold.	Silver.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Ounces Per Ton.	Ounces Per Ton.
Made without copper-ore.	61.	1.6	.....	2.18	7.1	9.7
Matte made with little copper-ore.....	68.5	3.5(a)	27.9	.....	4.2	8.7
Matte.....	.....	5.57	.....	.....	17.65	10.25
Typical matte.....	.....	22.6	30.0	.....	17.11	21.4
Typical matte.....	.....	20.6	.....	.....	11.73	19.7
Typical matte.....	.....	19.3	.....	.....	11.62	18.76

(a) The slag accompanying this matte had an assay value of $1.40 per ton.

The matte produced without copper-ore was made with the furnace running on pyritous material of the following composition: Iron, 24.52; silica, 27; lime, 3.06; lead, 5.82; zinc, 8.55; sulphur, 28.03; arsenic, 3.4 per cent.; copper, trace; gold, 0.04 oz., and silver, 2.86 oz. per ton.

It is worthy of note that, while some of the zinc enters the matte, practically no lead does. The analysis of the slag corresponding to this matte is given in Table I. under the name, "High alumina and zinc." The quantity of copper in this matte, 1.6 per cent., is insufficient to give a clean slag, which in this instance had a value in gold of $1.20 per ton. When operations were first started at the plant, an experiment was made of adding lead-ores in quantities equaling those of copper-ores now added, in order to ascertain whether lead would enter the matte and thus reduce the abnormal losses in the slag. However, no lead was found in the matte, for the reason that the conditions under which the furnace was operated precluded its entrance. There is so much oxidation that most of the lead becomes volatilized to the great detriment of the yield of silver. Lead, even in small quantities, is very undesirable in sulphide-smelting.

The first matte produced is generally resmelted twice, the third matte becoming the shipping-matte. Table IV. shows the concentration of metals in these three mattes.

TABLE IV.—*Concentration of Gold and Silver in Matte.*

Matte.	Copper.	Gold.	Silver.
	Per Cent.	Oz. Per Ton.	Oz. Per Ton.
First.....	13.5	4.04	6.03
Second.....	21.5	10.05	15.6
Third.....	22.6	17.11	21.4

The matte is cast into slabs in cast-iron molds, in order to break it up readily and have it in a convenient form for shipment.

Amount of Copper Necessary to Make Clean Slags.—At the present time, copper-ore for the matte is brought from Montana at a considerable expense, and is added in quantity just sufficient to produce the desired effect. It is aimed to have at least 10 lb. of copper in the charge for every ounce of gold present, and more, if a supply be available.

The silver is much more affected by the lack of a certain proportion of copper than is the gold. The ratio of silver to gold in the ore is from 1.5 to 2, to 1, and, with an equal saving of both metals, this ratio should be preserved in the matte. Mattes containing less than 20 per cent. of copper show a dis-

tinct loss of silver. In general, we do not think that sulphide-smelting is adapted to a close saving of the silver.

Fuel.—Owing to the high melting-point of the siliceous slags, the quantity of carbonaceous fuel used is considerable. The fuel expressed in percentage of total charge seems very high, but this explained by the very poor quality of the Cambria coke used, the ash amounting to about 30 per cent. Generally a mixture of two-thirds of Cambria coke and one-third of Eastern or Colorado coke is used. The quantity of coke used varies from 14 to 18 per cent. of the ores and fluxes charged into the furnace.

TABLE V.—*Composition of the Coke Used at the National Smelter.*

Kind of Coke.	Fixed Carbon.	Volatile Carbon.	Ash. (a)	Water.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
West Virginia	85.89	1.0	12.42	0.69
Cambria, Wyo	65.22	3.93	29.93	0.92
Colorado	86.86	1.7	10.7	1.6

(a) The analyses of the ash is given in Table II.

The statistics given in Table VI. show a capacity of the small furnace of about 105 tons of burden per day. The large furnace has a capacity of about 130 tons of burden per day. Both furnaces operate under the disadvantageous condition of treating a very large quantity of fines, aside from using such very poor and friable coke.

TABLE VI.—*Capacity of the Furnaces of the National Smelter.*

Material.	Smaller Furnace. November, 1903.	Smaller Furnace. December, 1903.
	Tons.	Tons.
Siliceous ore	1,100	980
Pyrite	512	548
Limestone	1,058	1,070
Copper-ore, etc	70	90
Flue-dust (a)	250	348
Matte	136	125
Total burden	3,126	3,161
Coke.		
Cambria	510	517
Eastern	180	162
Fuel percentage	18	17

(a) Including accumulated flue-dust from the chlorination works.

Sows.—In our opinion the production of sows is practically inseparable from sulphide-smelting when high concentration is done. Sows are due to the strong oxidizing effect of the furnace, as is shown by the following data: A desulphurization of 80 per cent.; the production of copper sulphate, found in layers in the accretions of the downtake; no evidence of carbon monoxide in the furnace-gases; the volatilization of all the lead fed into the furnace; and the facts that no iron goes into the furnace as oxide, and the slag contains from 18 to 20 per cent. of iron oxide in the form of silicate. These data make it difficult to imagine that the reducing conditions in the furnace could exist sufficiently strong to produce metallic iron.

We believe that the sows are produced by oxidation in a similar way that metallic copper is produced during bessemerizing, taking as the first stage, the melting of the pyrite, FeS_2, and the loss of the one atom of sulphur forming the monosulphide FeS; the second stage, the gradual oxidation of the sulphur in the monosulphide, producing a subsulphide; the third stage, the production of some ferrous oxide, part entering the slag, and part reacting with the sulphide present, producing sulphurous acid gas and metallic iron, according to the following chemical equation:

$$FeS + 2FeO = 3Fe + SO_2.$$

Experience has shown that a larger quantity and a higher pressure of blast result in an increased production of metallic-iron sow; and, from its analysis, it is seen that it contains practically no carbon, which apparently should be present if the metallic iron were due to the reducing action of the coke and the carbon monoxide.

In the large furnace (144 by 38 in. in cross-section at the tuyeres), a 15-ton sow was produced in a 7-months' run, having an approximate value in gold of $5,000. In the smaller furnace in a 3-months' run, under a lower blast-pressure, a 5-ton sow was produced, having an approximate value of $1,500.

The sow, as a whole, is not homogeneous, and consists mainly of metallic iron containing intermixed slag and a little matte. The metallic iron contains practically no silver, but considerable gold. The iron has a crystalline structure similar to that of pig-iron, possesses a distinct silver color and is practically

pure metallic iron. In the large sow some pieces of copper were found. Rarely a small button of lead is found in a sow.

TABLE VII.—*Composition of Sow Produced at the National Smelter.*

Material.	Iron.	Sulphur.	Gold.	Silver.	Copper.
	Per Cent.	Per Cent.	Oz. Per Ton.	Oz. Per Ton.	Per Cent.
Average sow	.....	.....	20.3	2.4	.....
Crystalline iron	99.68	0.2	29.6	4.4	trace
Fore-hearth sow (a)	.....	.....	35.8	nil	14.19
Lead in sow	.....	.....	134.17	273.45	.....
Copper in sow	.....	.....	83.8	nil	.....

(a) A true sow.

Material from the fore-hearth resembling a sow contained: Iron, 72.3; sulphur, 19.19; copper, 6.94 per cent.; gold, 6.3 oz., and silver, 5.2 oz. per ton. It is usually through the accumulation of this material and that of the true sow, that the fore-hearth is lost. The top layer of the sow, as well as that part in contact with the fire-brick, usually has an oxidized appearance.

The treatment of the sows for the recovery of the gold and silver values in them is a difficult problem, especially when no reverberatory furnaces are available. The National Smelter has not this valuable adjunct which is practically necessary for the treatment of the sows and the flue-dust. At the present time the sows are broken up by blasting, a very expensive operation, and re-fed into the furnace a little at a time with the pyrite in order to resulphurize the iron.

Flue-Dust.—Owing to the absence of reverberatory furnaces, the flue-dust made, amounting in quantity to about 10 per cent. of the charge, is resmelted in the blast-furnace—a rather undesirable procedure. Aside from this flue-dust, the plant has treated at times considerable quantities of concentrates from stamp-mills, and accumulated flue-dust from chlorination-plants, so that the quantity of fines was really more than the furnace could profitably handle. In order to treat this material and thereby keep within limits the quantity of flue-dust produced, the furnace, low in itself, was operated with a low charge, and consequently with a fairly hot top, which accentuated the losses in fume. The flue-dust increases in value as the stack is approached.

TABLE VIII.—*Analysis of Flue-Dust.*

Material.	Fe_2O_3.	SiO_2.	Lime.	Sulphur.	Copper.	Gold.	Silver.
	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.	Oz. Per Ton.	Oz. Per Ton.
From dust-chamber.....	.....	.....	.....	.....	0.88	1.02	4.1
Beginning of flue (*a*).....	37.14	35.7	4.12	5.60	0.80	1.17	12.60
Average flue-dust (*b*).....	25.43	34	.....	4.63 (*c*)	0.80	1.25	13.55

(*a*) Contains also 4.07 per cent. of Al_2O_3. (*b*) Contains also 9 per cent. of carbon.
(*c*) Considerable of which is soluble in water.

TABLE IX.—*Analysis of Accretions in the Dust-Chamber.*

Place.	Carbon.	Copper.	Gold.	Silver.
		Per Cent.	Oz. Per Ton.	Oz. Per Ton.
From the bottom of downtake.....	Considerable.	14.1	0.70	8.30
On the blast-heating apparatus.....	Considerable.	16.1 (*a*)	0.40	5.1

(*a*) All soluble in water, for the most part being present in the form of copper sulphate.

The ratio of gold to silver in the ore is usually about 1 to 2, so that the analysis of the accretions shows a heavy loss of silver by volatilization. In fact, the process as a whole is unfavorable to a high recovery of the silver, especially if high concentration is carried on.

Losses in Fume.—It has been the experience at the National Smelter that, under certain conditions, as, for instance, when operating with high concentration and lead or zinc in the charge, there is a considerable loss of values by smoke and fume, especially in silver, and gold. Lead is not at all desirable in the furnace, most of it being volatilized, and carrying values with it.

TABLE X.—*Analysis of Condensed Fume from the National Smelter.*

Place from Which Taken.	Gold.	Silver.	Copper.	Remarks.
	Oz. Per Ton.	Oz. Per Ton.	Per Cent.	
Flue near the stack.....	1.50	15.50	trace.	.....
Middle of flue.....	1.60	11.90	.....	.....
Near the beginning of the flue.....	1.60	10.40	.....	Contains lead, soluble sulphates, arsenic, some calcium sulphate.
Average value of fume from the steel roof of the flue.....	2.00	16.1	.....	

These analyses show the relatively much greater volatilization which the silver suffers, although it is evident that the gold in

the ore also suffers loss. Experience has shown that, when the furnace is running low with a hot top, on material containing some of the above-mentioned undesirable elements, the monthly account based on the ore-assays, will show a considerable loss in silver, a very appreciable one in gold, and some copper. Moreover, these losses are not slag losses. Greater attention paid to the saving of fume in sulphide-smelting plants will lead to economical working.

Origin of the Magnetic Iron-Ores of Iron County, Utah.

BY E. P. JENNINGS, SALT LAKE CITY, UTAH.

(Atlantic City Meeting, February, 1904.)

THE iron-ore deposits of Iron county, Utah, which rival those of Lake Superior in extent, are situated 275 miles south of Salt Lake City, and 25 miles south of Lund, a station on the San Pedro, Los Angeles & Salt Lake railroad.

The range of hills containing the deposits has a general NE–SW. strike, and rise from a few hundred to 2,000 ft. above the Escalante desert, the ore being included in an area of 5 by 20 miles.

The basal rock of these hills is diorite-porphyry. The flanks of the range are covered with limestone supposed to be of Carboniferous age, underlain in some localities by quartzite. Both formations are conformable to the porphyry, and are the remnants of beds that formerly covered the highest peaks, but have been removed subsequently by erosion.

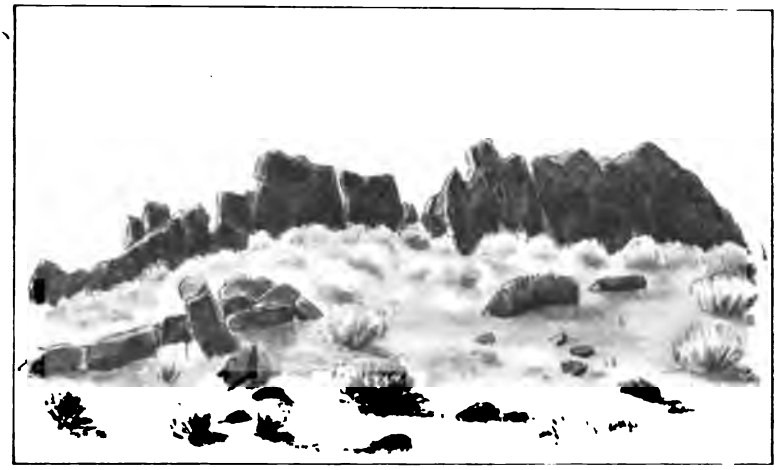
In the central and southern parts of the range, beds of Cretaceous sandstone rest unconformably on the eruptives.

Rhyolite-flows occur to the south and southeast of the iron-ore deposits; and dikes and sheets of post-Tertiary lava are common.

With the exception of a few springs, the range is destitute of water. Sage-brush, stunted cedars and the common desert-bushes constitute the only vegetation.

Large beds of coal of poor quality occur in Cretaceous sandstones 15 miles east of the ore-deposits.

FIG. 1.



THE GREAT WESTERN DIKE, SIDE VIEW.

FIG. 2.

THE GREAT WESTERN DIKE, END VIEW.

Both the magnetite and the hematite vary in quality from very pure ores high in iron and low in phosphorus and sulphur, to siliceous ores of no commercial value. The ores contain traces of copper, but no titanium. The composition of the better grades of magnetite is as follows: Fe, 62 to 68; P, 0.01 to 0.48; S, 0.007 to 0.03; and Cu, 0.002 to 0.08 per cent. The hematites yield on analyses: Fe, 40 to 61; P, 0.03 to 0.4; S, 0.01 to 0.1; and Cu, 0.003 to 0.09 per cent. The above analyses are from a large number made recently, and represent the variations in the merchantable ore.

The magnetite occurs in nearly vertical dikes in diorite-porphyry; and also as sheets in overflows from these dikes, very similar to the overflows of lava so common in Southern Utah.

In some instances the remnants of the sedimentary beds remaining in contact with the ore form one of the walls. The presence of these thin beds of limestone, or quartzite, has led a few observers to conclude that the ore is in the form of a contact-deposit, but, as a matter of fact, it is enclosed in syenite-walls below the sedimentary beds.

The dikes of hard, black magnetite form bold outcrops that are prominent features in the landscape, projecting their black masses like great walls above the crest of the light-brown porphyry-hills.

Some of the dikes are covered with an overflow sheet, and their exact position under the sheet is a matter of conjecture. As a rule, the original fissures are on the tops of the hills; and it may be assumed that the highest point of the overflow marks the position of the dike. In some cases the sheet covers both sides of the hill; and in these instances, it is evident that the hill-crest contains the original dike.

The typical dike-form, not complicated by accompanying sheet-structure, is shown in Figs. 1 and 2. One of the largest of these dikes is the Great Western near the northeastern end of the range, 10 miles west of Cedar City. It is nearly 900 ft. long and 30 ft. high. The surface of the hill below the dike is thickly covered with angular blocks of ore which have been broken from the outcrop.

The ore of all these outcrops is a massive, black magnetite without trace of crystallization, except in vugs and joints. It

is strongly magnetic; and natural magnets, or lodestones, are not uncommon. The chief impurities are silica in the form of quartz and chalcedony, and apatite, the latter occurring in slender hexagonal crystals, at times 6 in. long and 0.5 in. in diameter.

In the dikes the apatite is often symmetrically arranged in bands parallel to the walls, the individual crystals that compose these bands being perpendicular to the walls. In the sheets, or overflows from the dikes, the apatite crystals are not so distinct, but, when observed, are found to be approximately normal to the surface.

The ore in the sheets contains many small cavities lined with chalcedony, resembling in this respect surface-flows of lava. The ore of the dikes, however, is free from cavities.

Where the sheets are intruded between the porphyry and limestone, the magnetite has been converted into hematite, which differs entirely in physical character from the original magnetite. The hematite is of the soft, earthy variety, and is lower in iron- and higher in lime-content than the magnetite.

My conception of the origin of these magnetic iron-ores is, that they are the result of an extreme basic differentiation from a basaltic magma in the deep; and, in a molten condition, they have been forced to the surface through fissures that were formed contemporaneously with the eruption of the magnetite, in a manner similar to the formation of basaltic and other eruptive dikes. When the quantity of material was more than sufficient to fill the fractures, it overflowed in the form of sheets. When the fractures extended into the over-lying, sedimentary beds, they also have been filled with ore.

Erosion has destroyed most of the enclosing-walls of this character, but fragments of angular quartzite inclosed in the ore is sufficient proof that, at least, some of the upper parts of the dikes were contained in quartzite.

That these deposits were not simple segregations in place is evident from the fact that they extend into the sedimentary beds.

The diorite-porphyry is not sufficiently basic to have furnished iron for these enormous deposits by a process of segregation, but the basic lavas that are extensively developed in southern Utah may be another phase of the magma that was the source of the magnetite. More probably they have originated from the fusion of ancient ore-beds in the deep.

Professor Kemp, to whom thanks are due for his assistance, has determined the wall-rock to be diorite-porphyry. He has also examined thin sections of the ore for olivine, augite and plagioclase, with negative results. The sections examined by him showed considerable quartz, in the form of infiltrations. But, while these results are strongly against an eruptive origin for these ores, the evidence in the field is strongly in favor of it.

The Utah iron-fields will be of great importance in the future; and this paper has been prepared, in the hope that it will lead to a discussion of the method by which these great deposits have been formed.

Report of a Committee to Co-Operate in Standardizing Abbreviations, Symbols, Punctuation, Etc., in Technical Papers.

(Lake Superior Meeting, September, 1904.)

THIS Committee is the result of a desire of the authorities in charge of the publications of the four national engineering societies to co-operate in this matter.

The members of the Committee are the following:

CHARLES WARREN HUNT, Secretary of the American Society of Civil Engineers.

D. S. JACOBUS, Vice-President of the American Society of Mechanical Engineers.

JOSEPH STRUTHERS, Assistant Editor of the American Institute of Mining Engineers.

CARY T. HUTCHINSON, Chairman of The Editing Committee of the American Institute of Electrical Engineers.

This Committee has held several meetings; it seemed advisable, at the outset, to limit its discussions closely to the general subject of abbreviations. Further, it seemed best to formulate a few general rules to be followed in making abbreviations, rather than to compile a list of forms to be recommended.

The Committee decided to limit the subject more narrowly by considering only abbreviations to be used in the text, or general reading-matter, and not those to be used in special matter,

such as columns, box-headings, plates, figures, etc. The rules that follow are intended to apply to the text, and not primarily to such special matter. This Committee is of the opinion that it is impracticable to make general rules applicable to special matter; it believes that the rules herein stated should be followed as far as possible even in special matter, realizing, however, that clearness is of the first importance, and that all rules must be secondary to that consideration.

Referring, then, to abbreviations in the text or general reading matter, the Committee recommends the observance of the following rules:

1. Use abbreviations only after nouns denoting a definite quantity. Example: "The power plant has a capacity of 10 h.p.," not "10 horse power;" but, "The capacity of the plant, in horse power, is ten."

2. Do not abbreviate abstract or descriptive words. Example: "horizontal return tubular boilers," not "h.r.t. boilers."

3. Use lower-case characters for abbreviations. An exception to this rule may be made in the case of words spelled normally with a capital. Example: "B.t.u." and not "b.t.u." or "B.T.U." (British thermal unit), "U. S. gal." (United States gallon), "B. & S. gauge" (Brown and Sharp gauge).

4. Use a period after each abbreviation. In a compound abbreviation, do not use a space after the period. Example: "i.h.p." and not "i. h. p." (indicated horse power).

5. Use a hyphen to connect abbreviations in cases where the words would take a hyphen if written out in full. When a hyphen is used, omit the period immediately preceding the hyphen. Example: "3 kw-hr." and not "3 kw.-hr." (3 kilowatt-hours).

6. Use all abbreviations in the singular. Example: "17 lb." and not "17 lbs." (17 pounds), "14 in.," not "14 ins." (14 inches).

7. Never use "p." for "per," but spell out the word. Example: "100 ft-lb. per ton" (100 foot-pounds per ton); "60 miles per hr." (60 miles per hour).

8. Use decimals, as far as possible, in place of vulgar fractions. Example: "1.25 ft.," not "1¼ ft."

9. In general, spell out an adjective qualifying the name of a unit. Example: "boiler h.p." (boiler horse power). The

exceptions to this rule are—"i.h.p." (indicated horse power), "e.h.p." (electric horse power), "b.h.p." (brake horse power), "e.m.f." (electromotive force), "m.m.f." (magnetomotive force).

10. Use "Fig.," not "Figure." Example: "Fig. 3," and not "Figure 3."

11. In all decimal numbers having no units a cipher should be placed before the decimal point. Example: "0.32 lb.," not ".32 lb."

12. In the notation of large numbers, use "en" spaces instead of commas. Example: "1 520 125," not, "1,520,125."

13. Use the word "by" instead of "x" in giving dimensions. Example: "8 by 12 in.," not "8 x 12 in."

14. Never use the characters (′) or (″) to indicate either feet and inches; or minutes and seconds as periods of time.

The following forms are given as illustrations of these rules, and are recommended to be used:

Name.	Abbreviation.
Inches,	in.
Feet,	ft.
Yards,	yd.
Miles,	spell out.
Pounds,	lb.
Grains,	gr.
Tons,	spell out.
Gallons,	gal.
Metres,	m.
Millimetres,	mm.
Centimetres,	cm.
Kilometres,	km.
Kilogrammes,	kg.
Grammes,	g.
Milligrammes,	mg.
Kilogramme-metres,	kg-m.
Metre-kilogrammes,	m-kg.
Seconds,	sec.
Minutes,	min.
Hours,	hr.
Linear,	lin.
Square,	sq.
Cubic,	cu.
Per,	spell out.
Fahrenheit,	fahr.
Centigrade,	cent.
Per cent,	% or per cent.
Volts,	spell out.
Ohms,	spell out.
Watts,	spell out.

Name.	Abbreviation.
Kilowatts,	kw.
Kilowatt-hours,	kw-hr.
Watt-hours,	watt-hr.
Amperes,	spell out.
Brake horse power,	b.h.p.
Electric horse power,	e.h.p.
Indicated horse power,	i.h.p.
British thermal units,	B.t.u.
Gramme-calories,	g-cal.
Kilogramme-calories,	kg-cal.
Magnetomotive force,	m.m.f.
Electromotive force,	e.m.f.
Revolutions per minute,	rev. per min.
Circular mils,	cir. mils.
Miles per hour per second,	miles per hr. per sec.
Candle-powder,	c-p.
Watts per candle-power,	watts per c-p.
Mean effective pressure,	spell out.
High pressure cylinder,	spell out.
Diameter,	spell out.

The members of this Committee have agreed to recommend the authorities in charge of the publications of the four national engineering societies to follow the rules given herein in their respective publications, as far as possible.

(Signed) CHARLES WARREN HUNT,
D. S. JACOBUS,
JOSEPH STRUTHERS,
CARY T. HUTCHINSON.

NEW YORK, March 11, 1904.

SECRETARY'S NOTE.—The foregoing report, though informal and tentative, is published in this volume for the information of members, from whom criticism is invited. By the authority of the Council, all matters of style, punctuation, typography, etc., are left to the discretion of the Secretary, as Editor of the *Transactions* and other publications of the Institute; and in the use of this discretion, the Secretary has for many years followed a general system, sufficiently elastic to permit some concessions to the wishes of authors, in matters not essential. The desirability of uniform practice with regard to the points embraced in this report is recognized; and the Secretary, while reserving the privilege of individual action, is glad to say that the recommendations of this Committee of Consultation represent, in most particulars, his own practice hitherto, and that most of the recommendations of the Committee which

involve a departure from that practice, he would be willing, for the sake of uniformity, to adopt, if they should be adopted by the editors of the sister societies concerned. To this statement, he is disposed at present to make the following exceptions:

1. He is not willing to discard the use of commas in the notation of large numbers, as recommended in No. 12 of the Committee's suggestions. This does not refer to the arrangement of tabular matter, the consideration of which is disclaimed by the Committee. His reasons need not be amplified here; but the following among them may be briefly stated:

a. The comma would probably still be used, in the notation of numbers under 1,000,000; and the substitution of an "en" space in larger numbers would be an unnecessary employment of two ways of signifying the same thing.

b. The abandonment of this use of the comma is not called for, unless the comma is to be the "decimal-point," as in some foreign literatures. Since American usage employs the period as the "decimal-point," the present general use of the comma in notation need not be disturbed.

c. The attaching of significance to a simple space in printed matter, as proposed, is objectionable, because a space may be due to the dropping-out of a type—an accident which, as all proof-readers know, may happen to a "form" after the last editorial revision. It is true that the "en" space, specified by the Committee, is too narrow for an ordinary figure; but the difference between an "en" and an "em" space is easily overlooked by the printer, and even by a hard-pressed proof-reader. (In the proof of the above report, as it came before the Secretary, the printer had introduced at this point the larger space.) Now, for equal final accuracy, the system which makes the proof-reading easiest is to be preferred.

2. In the designation of thermometer-scales, the use of "fahr." for Fahrenheit, and "cent." for centigrade, as given in the Committee's list of illustrations, is objectionable. These terms fall within the exception stated by the Committee under No. 3. The three thermometer-scales are those of Fahrenheit, Celsius and Reaumur; and the abbreviations denoting them should begin with capital letters. Indeed, the capital letters, combined with the symbol indicating degrees (° F, ° C and ° R), are quite sufficient and unmistakable.—R. W. R.

Appraisal of the Value of Mineral-Lands, with Especial Reference to Coal-Lands.

BY H. M. CHANCE, PHILADELPHIA, PA.

(Lake Superior Meeting, September, 1904.)

In recent years an important function of the mining engineer has been the appraisal of the value of mining-properties required by those planning consolidations of a number of individual operations.

In no field of productive industry have the economic results attainable by consolidation been more clearly shown than in the mining-regions of the United States. In many important mining-centers unification of ownership of the most valuable properties has already partially or completely been attained, and, in other districts, projected absorptions and consolidations await only favorable financial conditions for their consummation. The iron-ore districts of the Lake Superior region, the coal-mines of Colorado, of Wyoming and of Utah, the anthracite mines of Pennsylvania, the Connellsville coking-coal mines, and large areas of bituminous coal-land in Pennsylvania, in Ohio, in the Indian Territory and in other portions of the United States, are now largely controlled by a few corporations operating on a large scale, or by a number of corporations working in harmony because of an affiliation and interleaving of interests.

The tact of keen financiers and the diplomacy of men of great administrative ability are the chief operative forces in conceiving and accomplishing such consolidations, but in formulating and maturing them, the services of the mining engineer are often required to supply the facts upon which they must be based. For this purpose the expert is expected to furnish full and complete reports covering the quantity of ore, coal or other mineral available, its value per ton, cost of mining and treatment, etc.

In reporting upon gold-, silver-, lead-, zinc-, copper- or iron-

property when the purchase of the property or its consolidation with other property is contemplated, the expert usually is expected to appraise the value of the property, if not directly at least indirectly, by estimating the quantity of ore actually in sight, and ore "partially in sight," determining its gross value per ton, and its net value at the smelter after deducting cost of mining, transportation, etc.

Many articles discussing the computation of the value of mines by estimation of "ore-in-sight" and other methods, are found in our *Transactions* and in other technical publications, but little has been written regarding the appraisal of the value of coal-lands.

The paucity of literature covering this field of work may be attributed to the fact that experts reporting upon coal-property, under past economic conditions, commonly have not been expected to include such an appraisal as a part of the report covering the property. The function of a mining engineer or geologist generally has been to examine undeveloped property; reporting to intending purchasers the thickness, quality, number and yield per acre of workable coal-beds, the purchaser determining whether the price asked for the property is satisfactory. In recent years, since the consolidation of developed properties has been in vogue, the engineer more frequently has been called upon for appraisals of the commercial value of going concerns, including the value of land, plant and improvements, and the good-will of the business.

In many cases this work is divided between the geologist, the consulting mining engineer, the constructing engineer and the expert accountant, each of whom separately or jointly is asked to report; the first upon the quantity of workable coal; the second upon the cost of mining, capacity of mines and value of improvements and developments; the third upon the value of plant and outside equipment; and the fourth upon the cost, return and net profits as shown by the operating company's books. These reports furnish material from which the financier may determine the relative values of the properties under consideration, and what sum they are fairly worth. To use this information intelligently requires sound business judgment, for the price asked by the vendors must always be greater than the aggregate value of land, plant and improve-

ments shown by these several reports, because it includes a sum representing the value of the "good-will" of the business, or its "position in the trade."

The engineer actively engaged in the development or management of coal-properties rarely has such relations with the financial management as qualify him to make commercial appraisals; and to a recognition of this fact must be attributed the practice of some financiers in relying upon their own judgment in reaching conclusions concerning the value of coal-property. Doubtless, in a majority of cases, the financier is the best judge of commercial value, and if supplied with the necessary data covering acreage, quality, yield per acre, cost of mining and past profits, his conclusions are generally correct. But that grave errors of judgment may occur through misinterpretation of the facts presented by the engineer, or through a failure to grasp important economic conditions, is evidenced by excessive prices sometimes paid for coal-property, and the consequent overcapitalization and bonding.

Therefore it is essential that the engineer should qualify himself for undertaking the commercial appraisal of coal-properties, because much work of this nature will be required in all of the important coal-mining districts before the consolidation of interests is complete in each district. To do this work efficiently he must add to his training as an engineer, a knowledge of the economic conditions affecting the cost of production and transportation, the element of competition, the market-possibilities, and something of the acumen of the trained man of business—"the man of affairs."

The purpose for which an appraisal of mineral property is desired will determine the choice of method or combination of methods to be used. If it be desired to ascertain the price that can be realized for the property and franchises of a corporation, if these be disjointed and separately sold, the method must differ from that employed when the object is to determine the real value of the properties to owners capable of profitably operating them. At the outset it is usually possible to eliminate from consideration the methods based upon forced-sale valuations, first, because such valuations are commonly untrue, erroneous and misleading; and second, because modern business methods rarely permit the owners or creditors of large properties to con-

sider disintegration, the course adopted almost invariably being to preserve the integrity of such properties unimpaired, because of the larger profit and greater value obtainable from them as a whole than could be realized, in the aggregate, from the several parts. Other methods have been applied to problems of this kind, among which the following may be noted:

1. A method occasionally used in the past is to determine the value by adding to the cost of the land, the cost of the improvements and a reasonable remuneration to the party which has successfully developed the property.

2. A method modeled after the common practice of real-estate appraisers is to determine the value by the prices at which property of similar character in the immediate neighborhood has recently been sold.

3. A method, ably elaborated by Mr. Joseph S. Harris several years ago for the purpose of appraising the value of coal-lands owned by the Philadelphia & Reading Coal & Iron Co., has been adopted by many experts for general purposes. By this method the total workable coal in the ground is first determined, and valued at a certain sum per ton, this estimate being based either upon what the coal would produce if leased upon a royalty, or upon the profits of mining it. Using as a basis the rate of increase in production, as shown by past experience, the probable yearly increase of output is calculated; and from these figures the probable revenue is calculated for each year of the period during which the assumed output can be maintained, or until all the coal is mined. Then the probable future earnings of the land, either by royalty or through operation, are capitalized at their present money-value, by the usual formulas for deferred payments, at a certain assumed rate of discount. In his report on the coal-lands above referred to, Mr. Harris clearly shows that the present money-value of coal-land depends largely upon the time at which development is to be commenced, the time elapsing before maximum output is attained, and the time to be occupied in exhausting the tract,—the present money-value decreasing rapidly as any of these variables is increased.

4. Another plan more generally used is to assume that the property can be operated at a certain yearly output, and can maintain this output for a fixed term of years at an average

profit per ton extending throughout the whole period, and not providing for any increase in output beyond what may be already in sight. The capitalized value of such yearly earnings is taken as the appraised value of the property.

5. In recent years it has become common to base the value upon the actual net earnings, allowing for such reasonable increase as the conditions of the trade indicate may be expected within one or two years, and for which improvements and plans have been projected and provided, treating the proposition from a business standpoint as a going and growing business, which fairly should be worth the price which the earnings justify, provided it be not grossly in excess of the appraised value of the land, plant and improvements, as reached by other methods.

The first method may be dismissed without serious consideration, because it is impossible to determine what would constitute a reasonable profit to the operator developing a tract of land, and, further, because this method ignores the value of the business that the operator has established and the enhancement of land-values due to the development of the property.

The second method is discarded for similar reasons, also because it fails to recognize the fact that the price paid for coal-property is a measure only of the value placed upon it by the vendor, who, if not in a position to operate it, may be willing to part with it for much less than its real value. In buying from original owners coal-operators rarely pay full prices, but almost invariably what they believe to be a small fractional part of the real value.

The third method is one most valuable for the purposes for which it is used by Mr. Harris, namely, as a basis upon which reorganization may be planned, and a new company financed. It may not be adapted for general use, because it is cumbersome, and also because it does not include allowances for the value of established trade and connections.

The fourth method is useful in a majority of cases, as corroborative of valuations reached by the fifth method.

The fifth method is based upon the actual earnings and the ability to maintain output at an increased rate for a long period. Should the value so reached be greatly in excess of the appraisal obtained by valuing the lands at what they fairly are worth, plus the value of plant and improvements, it is evident that the "good-will" or earning-capacity is too highly valued.

Earnings as a Measure of Value.

While it is unwise to appraise at a high valuation the goodwill of a purely commercial business, the value of a manufacturing industry producing staple products includes, as one of its most important factors, the established reputation of its products, its facilities for selling and distributing its output and the connections and friendly affiliations with consumers, dealers and transporting agencies that enable it successfully to retain its grasp upon the trade, to extend its operations and to expand and enlarge its business. In other words, an industry of this character can only be valued at its true worth by taking as a whole its property, plant, improvements, reputation of its products, its established trade, selling facilities and connections. Proper investigation having shown that the output of the concerns can be maintained for a long period, even at a rate of production in excess of present shipments, its value as a business proposition may be safely appraised, and in so doing it is customary to assume that the business, including plant and property, is reasonably worth a sum upon which the yearly net earnings will pay a satisfactory return.

In order to confirm the valuation determined by this method, separate appraisals in detail should be made showing the value of the lands, plant and improvements.

Value of Plant.

It is a comparatively simple matter to fix the value of plant and improvements, because the cost of the materials, machinery and supplies, together with the cost of transportation and erection, or installation, can readily be ascertained. It may, however, be proper to discuss the principles upon which the valuations should be based and those allowances which should be made for depreciation.

When the mine has a reasonably long life,—*i.e.*, is not approaching exhaustion,—the plant and improvements, if modern (adapted to economic operation) and in good repair (in condition to give many years' service, or until the property is exhausted), may fairly be appraised at first cost including the cost of erection, or at the present cost of replacement, because they are fairly worth this sum if the mine is to continue in op-

eration. If this principle be adopted, no deduction should be made to cover depreciation excepting on those parts of the plant and improvements which have deteriorated in efficiency, or are approaching a condition requiring considerable expenditures for repairs or renewals, or are so old that their usefulness will be of short duration.

Value of Development.

In assigning values to underground mine-development, the life of the mine, its daily tonnage-capacity and the quantity of coal tributary to it should be reviewed; the first cost of the improvements necessary for working the property, including shafts, air-shafts, slopes, etc., should be computed; and a valuation reached approximately equal to the cost of the development and improvements necessary to secure a like capacity under similar conditions at a new plant. This estimate should fairly represent the value to the owner of the underground development-work, but in cases where the life of the mine is short, a material deduction must necessarily be made from the appraised value.

In addition to the value placed upon underground development-work proper, an additional sum should be included to represent the value of development-work or dead work, that has been done in excess of that required to maintain the output of the mine at its rated capacity. This work usually consists of headings, or entries, driven into new territory from which no coal has yet been mined, and these should be appraised at their full cost. In addition, in some mines a large amount of heading- or entry-work exists from which the coal has partly been mined; in these cases the work may be appraised at a value proportioned to the quantity of available unmined coal tributary to them. Another item of value which should be included in an appraisal of the value of underground development is the special work provided for future requirements connected with ventilation, drainage, underground transportation and the opening of new territory by means of inside slopes, planes or shafts.

Value of Coal-Lands.

Coal-lands should be appraised at what they are fairly worth to the coal-operators, including the increment in value due to

the fact that the property has been opened and is being worked, but without considering the value of the good-will, or of the plant and improvements. A distinction must necessarily be made between lands already developed, those that can, or are likely to be, developed in the near future, and those the development of which may be postponed for many years.

No rules can be laid down for the determination of a fair value. The appraiser must review the prices at which similar property has been actually sold, the earning capacity, mining costs and profits, and the value placed upon the land by the owner, or, in other words, the least price at which the owner would be willing to sell. These factors must be carefully weighed and his final appraisal should represent an unbiased judgment of the real value.

In appraisals covering large areas in which the workable coal exists under greatly differing conditions of depth, accessibility, quality, etc., the territory should be sub-divided and separate appraisals made of each section. In this way deep territory is separated from that of moderate and shallow depths; drifting- is distinguished from shafting-territory; coal of specially good quality is appraised separately from that of moderately good quality; and thick coal-beds receive larger values than thin ones.

Value of Leaseholds.

In appraising the value of a leasehold it may be assumed that the lessee, having developed the property and being engaged in working it, can be expected to earn as profit a sum at least equal to the royalty paid for the coal. Of course this is a matter capable of being investigated by reference to the books of the operating company,—a proceeding which is rarely necessary, because the mining engineer should be able to determine whether the coal can be worked at this profit or not. Upon this basis the revenue derived by the operator would equal, and probably exceed, the revenue received by the lessor, and when this is true it is safe to assume that the right possessed by the lessee to mine the coal is equal in value to the vested right still remaining in the lessor. As it not infrequently happens in regions being actively worked, that the owners of leased property sell the lands to others subject to the operation of the

lease, a means is thus presented for determining the commercial value of the leasehold.

Attention may here be directed to the fact that this latter method of determining the value of the vested rights of the lessor and lessee in property operated under lease, indirectly offers another method for the appraisal of the value of lands, where the coal is owned in fee by the operator; for it may be assumed that, if operated under lease, the coal would command a certain royalty per ton, and the vested right of the lessor in the lease would be marketable at a fixed price per acre; further, that the vested right of the operator in the lease would be worth a certain price per acre. Merging these two interests into one, the value of the lands as owned in fee may be taken as equal to the sum of the values if operated under lease.

Factors Affecting the Value of Coal-Lands.

Before proceeding to an appraisal of coal-land values, the engineer must familiarize himself with the quality and quantity of coal available, the thickness of the workable beds, the cost of mining and of transportation, the mining conditions presented by the district, the character of improvements required, and other conditions affecting the industry as a whole, in the region concerned. In the following pages an attempt has been made to discuss briefly the principal factors which should be carefully investigated. While much of this information may appear unnecessary to the engineer, because the facts may seem self-evident, his clients, and others, may have merely a superficial knowledge of coal-mining, and may need enlightenment upon the most simple matters.

1. *Quality of the Coal.*—Coal of superior quality commands a higher price, assures a ready sale, retains the trade and insures a large output and a low mining-cost during times of depression. Coal of poorer quality rarely brings as high a price and is difficult to sell, especially when the trade is depressed; large commissions must often be paid for selling it, and the reduced shipments with irregular operation of mines, due to lack of orders, materially increase the mining-cost per ton.

The same considerations apply to coke. Therefore it is evident that quality is a factor of prime importance.

2. *Thickness of the Coal.*—To compete successfully with other

operators, it is essential that the beds to be worked should be as thick and as free from troublesome slate-partings, and that they should exist under conditions as favorable to cheap mining as the average present conditions at a majority of mines in the same district.

Other conditions being equal, the thicker coal-seams, and those containing little refuse, can be mined at a lower cost than those thinner or more impure. In each district, experience determines the proper thickness, character and quality necessary to constitute a workable coal-bed, that is, one that can be profitably mined.

It follows, as a matter of course, that in each district as the thicker and better beds become exhausted, and competition from them eliminated, the thinner and more impure beds will become workable.

In appraising the value of coal-property, it is customary to include only those coal-beds that are now, or shortly will be, workable; and to ignore the thinner and impure veins. For the reason that past experience has proved that in time the latter will become valuable, attention should be directed to this additional asset of large prospective but of indeterminate present value.

3. *Cost of Mining.*—The cost of mining is a most important element. It is not essential to success that the mining-cost at any operation should be low as compared with the region or State in which the mine is located; but it is important that the cost should not exceed the average of the district in which it is located. This statement is true because coal from each district usually has a definite market, *i.e.*, the market is limited to certain districts, certain railroads, or to a certain class of consumers who are best suited by the coal from that particular district; and coals from different geographic districts commonly do not compete in open market upon an equal basis, except where the transportation companies make compensating differences in freight-rates, in order to enable the miner to deliver coal from a district where the mining-cost is relatively high, at the price of more cheaply mined coal from other districts.

4. *Transportation.*—The question of transportation enters vitally into every discussion of values. Coal-land so located that it cannot be reached by railroad at a reasonable cost, or owned

by persons not commanding sufficient capital, or influence, to secure the building of a railroad, is of small value. Coal underlying farm-lands situated at a distance from existing railroads is of little value so long as the ownership remains vested in the farmer, or others unable to secure its development.

5. *Ownership.*—As already indicated, the value of a coal-property is affected by ownership. If the coal is owned by those able to mine it quickly, the value is greater; if it is to remain untouched for a long period, or to be worked on a small scale, the value is less. The concentration of large holdings into one corporation increases the value of the whole as a unit to a sum far greater than the sum of the values of the individual tracts, because a corporation can establish selling-agencies at all important distributing and consuming centers; can spend large sums for advertising; can form close affiliations with manufacturing and transportation companies; can retain able counsel and employ the best managers and representatives that can be found; can own its cars and can make whatever outlay may be necessary to build up a large and permanent business.

6. *Geographic Position.*—Under this heading it is sufficient to direct attention to the fact that those areas of workable coal, situated on existing lines of railway nearest the larger markets, must always be more valuable than territory remote from the large consuming centers. This fact is of especial importance with reference to lands situated upon any of the main trunk-lines of the United States, and to those near the great manufacturing centers of the country or most accessible to tide-water trade.

7. *Quantity of Workable Coal.*—It may seem rather paradoxical to assert that the larger the territory controlled by one corporation, the greater the value per acre of that territory, because attention has elsewhere been directed to the fact that the value of coal-property is greater where the coal can be quickly mined, than when its extraction (and the returns therefrom) must extend over a long period. This latter statement must be confined to properties of relatively small size, and is undoubtedly true of small properties. But the advantages accruing from the consolidation of ownership of large tracts under one management outweigh these considerations; because the large

control justifies extensive improvements and developments of a more permanent nature, and enables the owners to expend large sums in providing facilities for the upbuilding of a large trade, which necessitates extensive advertising, the establishment and maintenance of offices at all large distributing and consuming centers, and the employment of able managers for the conduct of the business. For these reasons the value of the coal-land within certain limits increases with the acreage controlled under one management.

8. *Mining Conditions.*—The factors affecting the cost of mining are—(1) actual cost of the mining operation; (2) possibility of planning large development with improvements of permanent nature; (3) output possible from each operation; (4) capital required for the plant and its development. These items depend upon many other conditions besides the thickness and purity of the coal; among which may be noted,—the depth at which the coal is found; the dip, pitch or slope of the bed; the quantity of water to be pumped, or the facilities for draining it away from the lands; the relative ease or difficulty of maintaining efficient ventilation; the character of the roof and the floor of the coal-bed; the hardness of the coal; the presence of gas in the mine; the character of the coal-dust (whether readily explosive or not); and the presence or absence of faults, rolls, and other disturbances affecting the regularity of the coal-bed.

It not infrequently happens that, while the quality and thickness of a coal-bed may be all that could be desired, other conditions may exist which render the bed almost, if not absolutely, unworkable under existing competitive conditions of other mines. These objectionable conditions are,—extreme depth or dip, troublesome faults or rolls, bad roof, soft floor, a great quantity of gas, a large quantity of water to be pumped, and troublesome dust.

9. *Character of Improvements.*—The value of any coal-property is affected by the cost of installing the plant necessary for its efficient development. If this plant be small, simple, quickly erected and inexpensive, other things being equal, the property will have greater value than if a large, complicated and expensive plant be necessary.

The character of the improvements needed depends some-

what upon the uses to which the coal is to be put. If it is to be sold for steaming-purposes (as run-of-mine coal) a very simple tipple is required. If it is to be sold in markets demanding screened coal, or if the vein contains objectionable impurities which must be removed, screening- and cleaning-devices must be installed, which require outside improvements of considerably larger cost than in the former case. Should the coal be especially adapted to coking, and the mines situated in a district where the economic conditions render it necessary or desirable, to transform the coal into coke, the erection of coke-ovens, and possibly also of cleaning- or washing-appliances, may be unavoidable.

10. *Market Reputation of the Coal.*—Under this head it is desired merely to recall the fact that the name of a coal-bed may be as valuable to the operator as is a trade-mark, brand, or copyrighted name, to the manufacturer of any well-known article or product. Furthermore, the mere location within the boundaries of some districts is an asset materially enhancing the value of coal-lands. This is a matter of great importance in the appraisal of coal-properties in the older districts, because it may enable the operator to find a ready market, and to derive large benefit from the established reputation of coal from other mines and from other coal-beds in the same district.

The Commercial Wet Lead-Assay.

BY H. A. GUESS, SILVERTON, COLORADO.

(Lake Superior Meeting, September, 1904.)

FOR a number of years I have used for the commercial wet assay of lead generally the ammonium molybdate, and occasionally the ferrocyanide method. These well-known methods need no detailed description here. In the ore-selling and ore-buying establishments of the West, 90 per cent. of all wet lead-assays are made by one or the other, and at least nine-tenths of this proportion by the molybdate method.

A procedure so well-established must have merit; and, in

fact, the ammonium molybdate method, when applied to siliceous ores or products of fairly high grade, has proved itself both rapid and accurate to a satisfactory degree. Yet those who use it most constantly clearly recognize its weaknesses in certain respects and under certain conditions; and I believe that a method free from such weaknesses would be welcomed by analysts. In this belief, I submit the results of my laboratory-experiments, covering a period of nine months, during which a method was developed by which more than 2,000 wet lead-assays were made on various ores and under purposely varied conditions, in order to determine the conditions necessary for accuracy, and to test the applicability of the method to all kinds of pure and impure lead-ores and products.

One weakness of the ammonium molybdate method is the end-reaction with the indicator—a freshly-prepared solution of tannin. The usual practice is to titrate at boiling-heat, and, for a high percentage of metal, to make a second boiling after getting a faint end-reaction, and then to finish to a complete end-tint. The correction for the indicator is, to some extent, affected by the personal equation involved in the operator's conception of a perceptible yellow tint, and varies among different analysts from 0.3 to 0.5 cc. of a standard solution of which 1 cc. equals 10 mg. of lead. It is evident that while this is quite permissible with fair-grade ores (and at the same time have different operators check quite closely enough for commercial purposes), it is an entirely different matter in dealing with tailings containing, say, 0.3 per cent. of lead, in which the variation in the correction used for the indicator would represent practically the total quantity of lead present.

A somewhat extreme instance showing this weakness came under my notice a few months ago. A series of samples of siliceous tailings, carrying from 0.3 to 0.5 per cent. of lead, were sent for checking-purposes to a leading umpire-assaying establishment in the West, with the statement that they were tailings for wet lead-assay. The report was, for about half of the lot, "trace;" for the remainder, "none." The analyst wrote us that the ammonium molybdate method had been used, and that, although he could see fine galena in the tailings, and could even pan it out, he had found, on assaying, that the number of drops of molybdate solution necessary to react with the tannin

was in each case no more than the correction-amount for the indicator; so that he was forced to make the report as he did.

Assuming that the amount of ammonium acetate used, and the bulk of solution for titration, were at the minimum limits for such fractional percentages, it is probable that if, in these cases, the analyst had given his flasks a prolonged boiling after adding the first few drops of molybdate solution, he would have found that his end-tint had faded sufficiently to accommodate a few additional drops of the standard solution. This would have given him, above the indicator-requirements, a small fraction which he could then have labeled "lead." A result, however, which has to be squeezed out by such labored and uncertain efforts is neither gratifying to the analyst nor really valuable to his client.

Furthermore, the lead molybdate precipitate being white, and the practice being common among analysts of not pouring the ammonium acetate solution through the sulphates on the filter, but of depositing filter and all in the original flask, digesting therein with ammonium acetate, and then titrating, it follows that, with only slight traces of lead present, there is no proof positive to the eye that the few drops of molybdate added really found any lead to precipitate, the solution being turbid from the slimes of the filter. The analyst is in doubt whether to report, say, 0.2 per cent. of lead, and chance it; to compromise on a "trace;" or to make a clean sweep and say "none." This hypothetical case may be considered by many as overdrawn, but I know from personal experience whereof I speak.

The other main and commonly-occurring weakness of the molybdate method is shown in dealing with ores containing large quantities of lime. Should the percentage of lead be fairly large, say 5 per cent. or more, the molybdate method does very well; but with small and fractional percentages of lead, all the above-cited troubles occur and are aggravated by the tedium of washing the bulky precipitate of pasty calcium sulphate derived from the sulphuric-acid evaporation. Ores of this class are common in southeastern Missouri; and, although remarkably pure in the sense of containing little else than galena in slightly siliceous dolomite, they have given much trouble to analysts who have to look for fractional percentages of lead in such material.

Some years ago, having experienced difficulties somewhat analogous to those above mentioned, I gave up the use of the molybdate method on material containing less than 1 per cent. of lead, and for some time, employed a method of precipitating those small amounts of lead from the hot, filtered ammonium acetate solution (after it had been acidified with hydrochloric acid), on a strip of pure aluminum free from silicon. The precipitated lead was removed from the aluminum strip by rubbing, washed, dried at 110° C., between filter-papers, and weighed as metallic lead. This method is tedious if many determinations are to be made, and it possesses several disadvantages, not the least being that, when the quantity of lead present exceeds from 20 to 30 mg., small portions of the lead-film are liable to become detached before the operation is completed, and, floating around in the acid solution, to be slowly dissolved.

The result of many trials led me to experiment with the precipitation of lead as chromate under various conditions; and this method, as finally elaborated and tested, has proved so rapid and satisfactory in every way that I have discarded all other methods, and use this not only for small percentages of lead, but for all wet lead-assays, on whatever material.

The insolubility of lead chromate in water, and in dilute acetic acid, is well-known, and the gravimetric method of estimation by chromate precipitation has been in use to some extent for years. Modifications, looking to the development of this method into a volumetric one, have also been published; but the fact remains that, at the present time, the volumetric chromate method as a commercial wet lead-method is practically unknown.

Taking advantage of the reactions of lead chromate, together with such literature as bore upon the subject, preliminary trials were made. A solution of normal potassium chromate was generally used, although dichromate answers nearly as well, the normal chromates being converted to dichromates on addition of acids; but as I have generally used the normal chromate for the precipitating solution, and as its equations are somewhat simpler, I will consider it here.

(1) $K_2CrO_4 + Pb(C_2H_3O_2)_2 = PbCrO_4 + 2KC_2H_3O_2.$

The potassium chromate solution is added in slight excess of the quantity necessary to precipitate all of the lead; the lead chromate is separated by filtration and washing; the filtrate containing the excess of chromate is acidified with about 25 cc. of dilute hydrochloric acid (1 : 1); a small crystal of potassium iodide (about 0.5 g.) is then added; and the liberated iodine is titrated with a standard solution of sodium hyposulphite in the usual manner, adding the "hypo" until the brown color of iodine has almost disappeared. Then a few drops of starch paste are added, and the titration is continued until the blue color has just disappeared, and a clear solution remains. The standard hyposulphite solution is preferably made of such a strength that 1 cc. will equal 0.5 cc. of chromate solution, in order to afford an easy calculation of the back-titration, in terms of chromate. The equations of the back-titration are:

(2) $2K_2CrO_4 + 6KI + 16HCl = 10KCl + 2CrCl_3 + 8H_2O + 3I_2$.
(3) $6Na_2S_2O_3.5H_2O + 3I_2 = 6NaI + 3Na_2S_4O_6$.

Equation (1) shows that, of a solution of K_2CrO_4 of 9.396, or approximately 9.4 g. per liter, 1 cc. will precipitate 10 mg. of lead. If the dichromate solution be used, it should contain 7.12 g. to the liter, in order to give the same strength. Equations (2) and (3) show that, of a solution of "hypo" of 36 g. of the pure crystallized salt to the liter (or 18 g. to the liter, if the more desirable half-strength be used), 1 cc. equals 1 cc. of the chromate standard. This is very nearly the strength of the standard "hypo" in common use for the iodide assay for copper, and if only a small number of wet lead-assays are made daily, it may be more convenient to use the copper "hypo" solution for the back-titration of the chromate; in the latter case the factor necessary to convert the copper "hypo" solution to terms of chromate is about 0.54, the exact factor being readily ascertainable by standardizing the one solution against the other.

Numerous trials were made by taking weighed quantities of pure lead sulphate, dissolving them in slightly acid ammonium acetate, diluting slightly, running-in a slight excess of the standard chromate solution, and, after filtering and washing, determining the excess by "hypo" titration. These tests have

shown that the precipitation of the lead is in strict quantitative accordance with Equation (1), so that, in preparing the chromate solution with pure salt, its theoretical value will be found to check exactly when standardized against pure lead sulphate under the conditions of the assay.

The accuracy of the standard having been determined against pure lead in solution as lead acetate, extended experiments were made to determine the possible interference of other elements, testing them in successive experiments by adding a different metal in the form of its convenient salt to each of a series of flasks, containing weighed quantities of pure lead sulphate; then digesting with nitric and sulphuric acids to complete white fumes; cooling; diluting with water; reheating in order to dissolve soluble sulphates; filtering; washing; returning the filter and its sulphates to the original flask; adding slightly acid ammonium acetate; and digesting a few minutes until the lead sulphate had completely passed into solution,—all in the usual manner of wet lead-assays. These respective solutions were then diluted slightly with cold water, and titrated with the standard chromate and "hypo" solutions, as described for the pure-lead standard.

These trials were made repeatedly, not only on artificially prepared charges, but later, on actual mixed ores, covering all conditions. The only possibly interfering elements are those which under these conditions do not go into solution readily as sulphates, and of these the calcium-barium-strontium series, tungsten and antimony are the only ones commonly occurring. Lime, although troublesome to wash, causes no interference in the results for lead, through any lime that might be in solution in acid ammonium acetate; since calcium chromate is freely soluble. Barium and strontium salts remain insoluble as sulphates, and tungsten yields the insoluble tungstic oxide, hence the compounds of these three elements cause no interference. Antimony, when present in considerable quantity, caused the results to be slightly low. This was found to be due to imperfect solution of the lead sulphate in slightly acid ammonium acetate in the presence of the residual mixture of antimony oxides, varying from Sb_2O_3 to Sb_2O_4 and Sb_2O_5, depending on conditions. This difficulty was rectified by prolonging the digestion of the filter and its contents, and using slightly am-

moniacal ammonium acetate for the digestion, in order to offset the acid nature of the antimony residues. The solution was then slightly acidified by the addition of acetic acid after dilution, and before adding the standard chromate. In none of these cases is it necessary after digesting with ammonium acetate to make a second filtration before adding the chromate; the flask with its contents and the remains of the filter, when cooled and diluted, are ready for the precipitation by chromate.

In adding the standard chromate solution, it is desirable, if the lead-content of the assay be known approximately, to add only a few cubic centimeters in excess, in order to minimize the washing and a lengthy back-titration. The filtration may be made directly after adding the chromate solution, using an ordinary 11-cm. filter of S. & S. No. 597, or No. 604, or any other fairly rapid and close paper; the lint present in the flask from the partial disintegration of the original sulphate filter, serving to check any tendency of the lead chromate to pass through. For the reason that the total bulk of the solution is only about two funnelfuls, and the precipitate one that is readily washed, this operation consumes but little time, and a 250-cc. flask serves to accommodate easily the filtrate, washings, and the hydrochloric acid necessary to liberate the iodine in the back-titration.

In dealing with fair percentages of lead, the exact quantities of ammonium acetate and of water for dilution are not of great importance; but when only about 1 per cent. of lead, or less, is present, and it is desired to make the filtration for the back-titration directly after adding the chromate, it is necessary to keep the bulk of the strong ammonium acetate solution used, within 10 cc., and to dilute with cold water to a bulk not exceeding 50 cc., before adding the chromate. As has been proved by careful experiments in this laboratory, in passing far beyond these limits, the solution seems to be on the one hand too strong in ammonium acetate for the small quantity of lead present, and, on the other hand, too dilute in milligrams of lead per cubic centimeter for the complete instantaneous precipitation of the lead chromate, although, if the time can be afforded, the precipitation becomes complete on standing a few hours, even when greatly beyond the limits given. From adding a large surplus of chromate solution, complete precipitation results at once, but such a surplus for back-titration is not desir-

able in dealing with small or fractional percentages of lead. If desired, however, an excess of chromate solution can be added, and, after filtering, instead of a back-titration (except in the presence of antimony), the funnel with its filter may be placed over a clean flask, and hot dilute HCl (1 : 1) added to the original precipitation-flask, in order to dissolve the lead chromate precipitate therein contained, taking care that the remains of the sulphate filter do not retain any undissolved precipitate. This hydrochloric acid is then poured through the funnel to dissolve the portion on its filter, washing both flask and filter with warm water. This hydrochloric acid solution of the lead chromate, after adding a little potassium iodide, is ready for a direct titration with "hypo" solution in the same manner as that for the back-titration, and the results are perfectly concordant with those calculated from the back-titration.

The bulky residues present from the large ore-charges usually used in low-grade assays render this modification tedious, owing to the time necessary in washing these residues free from the excess chromate solution, and it will be found more satisfactory to use the back-titration with the slight surplus of chromate solution.

Within the past few months I have had occasion to make several hundred wet lead-assays of ores and mill-products from southeastern Missouri of an average composition of about $CaCO_3$, 45; $MgCO_3$, 30; SiO_2, 8; Fe, 5; and lead, from 0.2 to 5 per cent. depending on the product. These assays were made in triplicate, by the chromate method above described, and very satisfactory accordance was obtained in each set. A triplicate series of the results obtained for the lead-content, taken at random from the assay-files, is given in Table I.

TABLE I.—*Results of Lead-Determinations by the Chromate Method.*

Material Assayed	I.	II.	III.
	Per Cent.	Per Cent.	Per Cent.
Slimes, Lot No. 2.....	3.13	3.16	3.13
Middlings.....	0.50	0.52	0.52
Tails.....	0.19	0.19	0.20
Concentrates.....	56.20	56.00	56.20
Concentrates.....	34.10	34.20	34.00
Middlings.....	0.62	0.64	0.64
Tailings.....	0.22	0.24	0.24
Slimes.....	3.88	3.94	3.90

In practice the method takes about the same length of time as the molybdate method on the same type of ore.

After making some scores of assays in this manner on these heavy lime-ores, it occurred to me that possibly the sulphuric acid feature might be omitted entirely, thus avoiding the tedious washing of the resultant bulky masses of calcium sulphate, without affecting the accuracy of the results. Experiments resulted in the following short-cut method for heavy lime-ores. To the ore-charge of from 1 to 5 g. in a 250-cc. flask add from 3 to 5 cc. of strong nitric acid and 15 cc. of strong hydrochloric acid; digest until everything is in solution, and the excess of acid has been reduced to about 8 cc. The whole operation on the hot plate requires but 15 minutes. The flask is then removed, and slightly dilute ammonia is added slowly in small excess, the neutralizing action being sufficiently vigorous to render the contents of the flask quite hot. Acetic acid of 80 per cent. strength is then added slowly, the flask being shaken vigorously, until its smell indicates a decided excess of acetic acid. Then 5 cc. of strong ammonium acetate is added to insure the solution of any lead compounds remaining undissolved by the ammonium acetate already formed in the flask.

If the ore contains no antimony, or separated gelatinous silica, and if the siliceous residue in the bottom of the flask is only in slight amount (as is usual with heavy lime-ores or with concentrates), add to the hot, undiluted and unfiltered solution, an excess—about 10 cc.—of a 10 per cent. chromate solution. Under these conditions, the bulk of the contents of the flask will not exceed 50 cc.; and, after shaking and letting the precipitated lead chromate settle for about 5 min., the contents are filtered through a 11-cm. filter of any fairly rapid and close paper. If these directions are carried out, the lead chromate will be quite granular, and will show no tendency to run through. The precipitate in the flask and on the filter is washed several times with hot water containing about 0.5 per cent. of acetic acid until free from soluble chromates. The funnel with its filter is then set over the original flask, and hot dilute hydrochloric acid (1:1) poured through the filter, dissolving the lead chromate. Further additions of hydrochloric acid are made if necessary until all lead chromate is dissolved

from the filter; then it is washed with warm water until free from chromate.

The original flask now contains nothing but the hydrochloric acid solution of the lead chromate and the washings, which, after adding a small crystal from 0.5 to 2 g. in weight of potassium iodide, is titrated direct with standard "hypo" solution whose value is known in terms of lead, the most suitable strength being that in which 1 cc. is equal to 5 mg. of lead. In this operation, by using only a small quantity of potassium iodide and having the solution fairly strong with hydrochloric acid (about 50 cc. of hydrochloric acid, 1 : 1, in a total of 200 cc. of solution) and somewhat warm, any tendency of the lead to form yellow scales of lead iodide, and thus somewhat obscure the end-reaction with starch, is completely checked, and the end-reaction has the same sharpness as in the back-titration of the originally described sulphate-chromate method.

This modification of the chromate method, as will be seen, does not consume, for heavy lime-ores, more than one-half the time of the sulphate-chromate method. It was so attractive that extended trials were made to see if its results in daily practice were concordant with the sulphate-chromate method which had already been thoroughly tested. Several hundred wet lead-assays were run by this method in parallel with the sulphate-chromate method not only on lime-ores, but on siliceous-ores, tailings, concentrates and artificially prepared charges, containing various additions of possibly interfering elements. It is needless to give details of all the tests. Concerning the influence of such possibly interfering elements, the results, verified by repeated trials, are as follows:

1. No interference is caused by the presence of aluminum, iron, arsenic, calcium, magnesium, copper, cadmium, zinc, uranium, phosphorus, tungsten or vanadium, under the usual conditions of the assay, as previously described; the results being concordant with those of the sulphate-chromate method.

2. Manganese causes no interference, but remains in solution, provided the precaution is taken to have the digestion of the ore made with sufficient hydrochloric acid in the proper ratio to nitric acid in order to insure that all of the manganese will be transformed to chloride.

3. Barium, if it passed into solution, would cause high re-

sults, because barium chromate is insoluble; but this element usually occurs as sulphate and, under the conditions of the assay, it remains insoluble and is therefore unaffected by chromate solution. Even if it occurred as a carbonate, there are usually sufficient sulphates formed from the oxidation of sulphides to cause it to separate out as the sulphate salt. However, to insure this reaction, it is only necessary in dealing with ores containing barium, to add 1 or 2 cc. of a 10-per cent. solution of ammonium sulphate, along with the usual addition of ammonium-acetate. The same considerations apply to strontium salts, should they be present in the ore, with this difference, that even if some strontium did pass into solution, it would only be precipitated as chromate from a highly concentrated solution.

4. Bismuth, on the addition of ammonia, forms a hydrate which, if in considerable amount, does not clarify completely with the addition of acetic acid; and, although this precipitate, on solution in hydrochloric acid, causes no reaction with potassium iodide, it is more or less bulky if present in large quantity, and it is troublesome mechanically in washing the precipitate of lead chromate. Therefore, in the presence of large amounts of bismuth, the sulphate-chromate method is the more rapid.

5. Antimony forms a residue of variable composition, which, if brought on the filter along with the precipitate of lead chromate and dissolved by hydrochloric acid, reacts strongly with potassium iodide and renders the assay unreliable, so that in the presence of antimony the sulphate-chromate method, which is in this case perfectly satisfactory, must be used.

6. Silver forms an insoluble silver chromate, which would be brought on the filter along with the lead chromate, and thus increase the weight of the latter by the quantity of the former present; and if this quantity is more than negligible it would entail a correction for silver, or would call for the use of the sulphate-chromate method.

As a summary of the foregoing, it may be said:

I. The sulphate-chromate method is applicable to the accurate determination of lead in all quantities, and in the presence of all commonly-occurring elements.

II. The modification of this method, by omitting sulphuric acid and bringing the ore into solution in nitro-hydrochloric

acid, with the precautions already cited, is applicable, except in the presence of antimony, or in the presence of considerable amounts of bismuth or silver; and it is a decided short-cut in point of time for ores containing a high percentage of lime. For siliceous ores and products leaving so large a residue, or separation of gelatinous silica, by the initial procedure of this method as to render a preliminary filtration advisable before adding the chromate, the bulk of solution produced by this operation and its attendant washings becomes so large that, on adding the chromate solution, the lead chromate, except on standing some time, does not separate out in a form sufficiently granular to remain completely on the filter, but tends to run through; in such cases the sulphate-chromate method will be found more satisfactory and more expeditious.

The screen-test on the tailings from a heavy lime-ore (Table II.) gives a good idea of the class of work that can be done on small percentages by the chromate method. The lead-value, as given for each screen-size, represents in each case the mean of three determinations, and the greatest variation in any instance was less than 0.03 per cent. Two of these determinations were made by the sulphate-chromate method, and one by the modified, or short-cut, method. The assay-value of the original tailings was 0.55 per cent., and the separate assays of the screen-sizes, when afterwards multiplied by their respective weight-percentages, amounted to approximately 0.57 per cent.

TABLE II.—*Tests of Tailings from Dry Screenings.*

Product.	Weight.	Weight.	Wet Lead.	Product of Screen-Size.
Original tailings.....	Grams. 2,000	Per Cent.	Per Cent. 0.55	Per Cent. .....
On 10-mesh.....	302	15.1	0.28	0.0422
On 20-mesh.....	722	36.1	0.30	0.1083
On 40-mesh.....	398	19.9	0.34	0.0676
On 60-mesh.....	140	7.0	0.41	0.0287
On 80-mesh.....	25	1.25	0.45	0.0056
On 100-mesh.....	54	2.70	0.76	0.0205
On 120-mesh.....	59	2.95	0.74	0.0218
Through 120-mesh.....	300	15.0	1.82	0.2730
Total.....	2,000	100.0	.....	0.5677

A wet method capable in ordinary routine-work of yielding such a concordance in fractional lead-percentages is, it would seem, worthy of consideration on the part of analysts.

Postscript.—Since working out the above sulphate-chromate method and the modified, or short-cut, method, I have learned that, although never introduced to any extent into commercial practice, a method somewhat analogous to the former was devised some years ago,[1] and, although my own work was quite independent, I wish all merit of originality to be placed with the earlier investigators where it belongs. So far as I am aware, however, the modified or short method, which is particularly applicable to low-grade, heavy lime-ores, for which the molybdate method is especially weak, has never been previously described.

Superficial Blackening and Discoloration of Rocks Especially in Desert Regions.*

BY WILLIAM P. BLAKE, TUCSON, ARIZONA.

(Lake Superior Meeting, September, 1904.)

TRAVELERS in the desert regions of the southwestern portion of United States, especially along the valley of the Colorado of the West, can hardly fail to note that most of the rocky outcrops of the harder rocks have a uniform prevailing blackness of the surface, in strong contrast with the normal color of the interior of the rock when freshly broken. For example, the granitic or gneissic rocks of Pilot Knob, below Yuma, are blackened beyond easy recognition. So also rocks upon the lower Mojave in California are so thoroughly blackened, as if painted, that ordinary gneiss can scarcely be recognized without breaking it so as to show the interior color and structure. The discoloration is not confined to the rocky outcrops in place, but is found upon boulders and upon the loose rounded pebbles which strew the surface of the mesas along the Colorado river. These fragmentary rocks partake of the general blackening.

[1] I. C. Bull, *School of Mines Quarterly*, July, 1902, p. 359–366.

* The substance of this paper was orally communicated in outline to the Cosmos Club of Tucson, Arizona, November 14, 1903.

They also have the smoothness of surface which characterizes all the rock subjected to the attrition of wind-driven sand and dust. They appear to be polished or lacquered. These phenomena are more particularly described in my report of a geological reconnaissance in California made for the general government in 1853.[1]

The coloration of rocks in tropical America did not escape the observation of Humboldt. In his noted work[2] he says that on the Orinoco "all blocks of granite, even pieces of white quartz, wherever they came in contact with the water, acquire a grayish-black coating, which does not penetrate 0.01 of a line into the interior of the rock. The traveler might suppose that he was looking at basalt or fossils, colored with graphite."

He also says that something perfectly analogous was observed by Rozier in the syenitic rocks of the Nile (near Syene and Philæ); also by the unfortunate Captain Tuckey, on the rocky banks of the Saire; and by Sir Robert Schomburk, at Berbice.

The question arises whence came this coloring; by what process has nature painted the rocks, covering them as with a coat of laquer or varnish?

Humboldt, finding such coatings upon the Orinoco and not along some other rivers where the waters were discolored or made dark brown by organic matter, appears to have been inclined to the opinion that the coloring was due to some substance of organic origin, peculiar to that river water, but writes that the phenomenon had not been thoroughly investigated, and it does not appear that any such investigation has yet been made.

The phenomena to be appreciated need to be seen over the broad areas where they are displayed. Hand-specimens fail to show the lustrous, even brilliant, character of the surfaces under the generally unclouded sun of our deserts of the Southwest, particularly those of Arizona.

The color and polish may be found upon nearly all the rock-croppings of the lower Gila and of the Colorado, particularly in Yuma county, so, also, in Santa Cruz county and in portions

[1] *Report of a Geological Reconnaissance in California*, 4to, N. Y., 1857, p. 231; also in *Pacific Railroad Explorations and Surveys*, vol. v.

[2] *Views of Nature*, by Humboldt. Bohn's edition, 1850, p 141; cited in *Geological Reconnaissance of California*, p. 231.

of Pima county, notably on the basaltic rocks west of Tucson where the Carnegie Desert Laboratory is established. But the glittering pebbles of the desert mesas give the most shining and impressive examples of all.

So far as investigation has extended, it appears that the coloration proceeds from within the rock, rather than from without, and that it is the result of chemical changes in the rock. It is endogenetic, rather than exogenetic, but we must consider also the influence of the atmosphere and of the sun's rays, which are no doubt powerful factors in the production of the film. The coloration is stronger on the upper and fully exposed portions of the pebbles, and is less distinct on the under side where partly buried in the earth and, as might be expected, such protected surfaces lack the polish of the exposed portions. As a rule, the blackened surface of the rocks does not extend below the soil. The portions protected by the earth from the light and outer air retain the natural tint or color of the rock, while the exposed portions are discolored. For example, the loose masses of basaltic lava used for building foundations at Tucson, Arizona, are blackened where exposed and are gray where they have been soil-covered. In the case of the pebbles the great difference in the color is perhaps best shown by white quartz which retains its normal whiteness below, but is colored above.

A plutonic rock from near Patagonia, Santa Cruz county, Arizona, is another example. It has an unusually black and lustrous surface due to a film not thicker than ordinary writing paper, but shading into a discolored portion to the depth of a millimeter or more, parallel with the irregularities of the weathered surface. In some places this discolored portion is duplicated by a second parallel layer or region of discoloration, both discolored areas giving evidence of chemical changes.

The dark-colored amygdaloidal basalt of the mountain west of the city of Tucson, already referred to, clearly shows by the difference of color between the surface and the interior, the result of alteration by chemical change to a depth of one-quarter of an inch in some places. While the surface is a very dark-brown, or black, the interior is a dark-gray, which near to the surface, or for a distance of 2 or 3 mm., is lighter in color than the mass of the rock and appears to have lost ferrous iron oxide

by its concentration at the surface where it has been oxidized to ferric iron.

The chemical examination of the outer film of such discolored rocks generally shows the composition to be iron sesquioxide. In some specimens manganese oxide predominates.

All the phenomena indicate that the coloration is due to the formation of a thin coating of ferric iron oxide or of manganese oxide, or of both, or of a mixture of iron oxides, such as magnetite, derived from the interior of the rock by an osmotic flow, a kind of rock-transpiration tending upwards and outward to supply the excessive evaporation under hot arid conditions. Such a flow, though imperceptible, is no doubt general in arid regions where the evaporation greatly exceeds the local supply by rain. Its results are noted on a broad scale in the formation of *caliche* and in the incrustation of salt and of other saline soluble compounds generally called "alkali" or "*tequisquita.*" In the case of rock-coloration it would appear that, in general, ferrous iron salts are changed to the insoluble ferric form under the influence of the sun's rays. In further support of this view it should be noted that most of the coated rocks contained originally, if not at the present time, iron in the ferrous form, either in biotite-mica, hornblende or other silicates, or as pyrite.[3] The metallic solutions may have originated at a considerable distance and depth from the place of final concentration and deposition.

While this explanation of the chemical endogenetic origin of the colored film is well sustained by the facts, we should not disregard the possibility of external factors other than heat and light, such as solutions in rain- or river-water. It can easily be conceived that the brown water of the Orinoco and other streams contains substances which would act upon solutions encountered in the pores of rocks, or presented at the surface. The action of a trace of tannic acid, for example, meeting traces of ferrous solutions in the rocks, would give, in the course of time, an inky blackness to the surface. We are not without an example of this nature. In the lead- and zinc-region of

[3] It should here be noted that in Arizona, as in New Mexico and other extremely dry regions, the outcrops of gossan accompanying mineral veins, notably of copper, are generally in the form of hematite, or of magnetite, rather than of limonite, which is common in moist climates.

Wisconsin piles of light-colored rock containing decomposing pyrite, when thrown out under oak trees, are found to be stained black as if by ink, the drip during rains carrying down from the oaks sufficient tannic acid to form an ink with the diffused iron sulphate.

In the case of the loose pebbles of the Colorado mesas, the endogenetic origin of the superficial coloring is not so certain or so easily explained. We should expect to find a greater diversity of coloring than actually exists if we consider the mass of the pebble alone as the source of the flow, and we would also expect to find a variation of coloring and of intensity corresponding to the varying degrees of hardness and of composition which such an assemblage of diverse rocks present. The mass of each pebble is not sufficient to yield an amount of coloring solution adequate to produce the change. This difficulty in the explanation may be removed by the assumption that there is an osmotic flow from the subjacent earth to the pebbles and that the solutions though small in volume and weak in composition are uniform in nature and are finally concentrated at the exposed surfaces. These conditions are partly suggested by the fact that pebbles of quartz, white inside, are found coated with a thin highly colored pellicle having a resinous oleaginous appearance which is not soluble in either boiling water or alcohol, but is readily removed by hot hydrochloric acid, with the formation of iron chloride.

The phenomena herein considered, and those of vadose circulation and the formation of mineral veins, are cognate and mutually explanatory.

The Investigation of Alaska's Mineral Wealth.*

BY ALFRED H. BROOKS.

(Lake Superior Meeting, September, 1904.)

THE developments of the past five years have shown that Alaska, as a field for mining, stands in the first rank among the possessions of the United States. Its annual gold output is now about $8,000,000. It produces silver, copper and coal in commercial quantities and its recently discovered tin and petroleum promise to become important products. Concurrent with the gradual development of this wealth, the mining public has ceased to regard the territory simply as an arctic province where a few placer-miners struggle with adverse conditions to secure a grub-stake or a modest fortune. Of late years there has been a large influx of capital to investigate its mineral resources, but in its area of nearly 600,000 sq. miles there still remain large, unexploited and little-known fields.

This work of investigation has been carried on under adverse conditions, and reflects credit on all who have shared in it, whether prospectors, mining engineers, or geologists. It is the purpose of this paper to outline briefly what has been, and is being, accomplished toward furthering the knowledge of Alaska's mineral wealth, and in this connection to present a brief history of the operations of the United States Geological Survey in the territory. A list of the publications of the Geological Survey, which pertain to Alaska, is presented as an appendix, in the hope that it may prove useful to those who are interested in the mining of this northern province.

During the Russian occupation of Alaska, from 1783 to 1866, but little attempt was made to delineate its geographic features or to study its geology and mineral resources. The Russian-American Co., which long held control of the territory, was entirely absorbed in the exploitation of its fur trade,

* Published by permission of the Director, U. S. Geological Survey.

without concern for its other possibilities. During the last decades of the Russian régime, however, the fur company was compelled by royal ukase to choose its administrative head from among naval officers, and some of these governors, notably Wrangell and Tebenkof, were men of scientific attainments and interests, under whose administration the finances suffered somewhat, but the cause of exploration was materially advanced by attempts at charting the coastal region, and even by an occasional expedition into the interior. From these surveys, supplemented by the efforts of navigators and explorers of various nationalities—chiefly English—was obtained a fairly complete knowledge of the entire coast-line of Alaska, and some geographic data concerning the lower courses of the Yukon, Kuskokwim and Copper rivers. Many of the expeditions included naturalists who made geologic observations and palæontologic collections, which eventually found their way to various scientific institutions of Europe; but, at best, the results were merely fragmentary. It was the policy of the Russian-American Co. to discourage the development of any mining interests within the territory, out of a belief that mining would be inimical to the fur-trading interest. When, however, the administration of the naval governors, combined with the decrease of the fur-bearing animals, to bring about a diminution of revenues, an abortive search was made for mineral wealth. A mining engineer by the name of Doroshin was dispatched to Sitka in 1848, and spent the succeeding two years in examining the islands thereabouts and in the Cook Inlet region; and he did, in fact, make the first discovery of gold in Alaska. But as the returns from nearly a year's work of about 40 men yielded only a few ounces of placer-gold, obtained near Cook inlet, he reported adversely as to the presence of gold in commercial quantities, thus terminating the only Russian attempt at gold-mining in Alaska. The current reports of evidences of Russian placer-operations in various mining-districts have so far proved to be without foundation.

Doroshin did, however, recommend the mining of coal at Cook inlet, and the demand created by the gold discoveries in California led to the establishment of such an enterprise on Port Graham, in 1852, but without commercial success. The Russians made no other attempt at developing mineral re-

sources, unless the shipment of ice to California, carried on in a desultory manner for some years, can be regarded as such.

The Russian traders had pushed their way a thousand miles up the Yukon, and had explored the lower stretches of the Kuskokwim and Copper rivers; but previous to 1865 little was known of the interior of what was then called Russian America. In that year it was reported at Sitka that American prospectors had found gold on the Stikine river, and an expedition was dispatched to verify the rumor and establish the International Boundary. With the party went Professor W. P. Blake as geologist, among the first of Americans to investigate the geology of Alaska. From the east the interior had been penetrated by the traders of the Hudson Bay Co., who, following the routes marked out by such explorers as Franklin and Mackenzie, had reached the Yukon in 1849; but they, too, were adverse to the development of mining industries and made no attempt to investigate mineral resources. In 1865 the Western Union Telegraph Co. organized a survey of northwestern America to find a route through Canada to Bering strait, in order to connect by cable with a line crossing Siberia from Europe. The success of the Atlantic cable led to the abandonment of the project, but the investigations of the scientists and explorers of the expedition were of lasting importance; though, except for the work of Dr. Wm. H. Dall, the most prominent member of the parties, they were of solely geographic interest. Dall continued his work in Alaska as a member of the Coast Survey after the transfer of the territory.

In 1866 Russia ceded her North American possessions to the United States for the sum of $7,200,000, and Russian America became Alaska. The interest awakened by the addition of this vast domain was only short-lived among the mass of the American people, the majority of whom regarded it as an arctic province possessing no value outside of the fur trade. Few, if any, dreamed that this isolated possession was destined to become a great mineral producer, and that in the course of one generation its annual gold output would exceed by over a million dollars the price paid for the entire territory.

The apathy of the public and the neglect of the executive and legislative arms of the government relegated the new possession to temporary oblivion. For 20 years after Alaska had passed

under the control of the United States, systematic surveys were limited to its coast-line. Explorations in the interior were fostered by the various branches of the government and by private enterprise, but only intermittently and without definite purpose. Thus, while there developed gradually a somewhat more exact geographic knowledge of our new territory, no light was thrown on its mineral resources. But while the government's interest in this virgin field lay dormant it soon attracted the ever-active American prospector. As already noted, he found gold on the Stikine even before the purchase of Alaska. Following this came the discovery of auriferous quartz near Sitka in 1879, and of gold in the Juneau placers in 1880. It was the development of the latter which led to the finding of the gold-bearing lodes that have made Juneau the foremost mining-camp of Alaska. The restless pioneers, soon finding means to overcome the opposition of the natives to the white men's penetrating the interior, made their way across the Chilkoot pass, and while descending the Lewes river came upon more gold. In 1886 the gold-placers of the Forty-Mile region were discovered, and the discovery of other districts followed rapidly during the succeeding decade.

These Yukon pioneer miners were dependent entirely upon their own resources, formulated and executed their own laws, and were practically ignored by the territorial and federal governments. It is largely to their perseverance and pluck that the country now owes its millions of revenues from the Alaskan placer-fields. Not until 1895 did Congress awake to the importance of examining into Alaska's mineral wealth. In that year $5,000 was appropriated for an investigation of its coal- and gold-resources by the U. S. Geological Survey; and, small as the sum was,—considering that the territory to be investigated was two and one-half times the size of Texas and that the field of investigation lay two or three weeks' journey from Pacific Coast ports,—this was the first organized attempt in this direction.

The party which was sent out spent the summer of 1895 in an extensive examination of the Pacific Coastal belt, and accumulated valuable information regarding the distribution of the coal-bearing rocks and the character of the gold-deposits.

In the following year a similar appropriation was made, and a party of three was sent inland to study the placer-districts

along the Yukon river. This long journey, at a time when few had penetrated to the Yukon gold-fields, was accomplished at the expense of considerable hardship, the party succeeding in visiting all of the producing gold-camps of the Yukon, among which were numbered at that time Forty-Mile, Birch creek and Minook districts. The investigation could not be continued in 1897 because the appropriation-bill failed to pass in time to make the funds available in this distant province. It was the discovery of the Klondike gold-placers in 1896 that opened people's eyes to Alaska's importance, and, as the facts became disseminated during the following two years, public opinion on this point rapidly changed. There was then pressing demand for reliable information about the northwestern part of the continent. This was met on the part of Congress in 1898 by appropriations for various bureaus for Alaskan investigations to be made by the Government; and the amount to be expended by the U. S. Geological Survey for this purpose was increased to $20,000. Under this latter appropriation the U. S. Geological Survey began the series of systematic surveys in Alaska, which it has continued to the present time.

The task before it was not an easy one. Here was an area of from 500,000 to 600,000 sq. miles, of which little but the coast had been surveyed, and very large areas were almost unexplored. The province to be investigated was far distant, and the season of operations limited to the summer months. Moreover, the work must be so conducted that the results should be available at the earliest possible date. No plans which required more than one season for their execution could be considered, for there was an urgent demand, on the part of the thousands who were working blindly in this northern region, for immediate information. It is plain that these conditions could not be met by planning detailed and final surveys, which, while of the utmost importance to a mining community, must be preceded by explorations and reconnaissances. Moreover, the fact that the work had to be begun within a month after the passage of the appropriation-bill, left little time to formulate plans and organize parties. Fortunately, the U. S. Geological Survey was able to draw on a corps of geologists and topographers who had been trained in other fields for work of this character.

It was decided to send four parties to Alaska: One to map an area lying close to the new Klondike placer-fields, and the others to conduct extensive explorations. One of the latter made a reconnaissance of the Sushitna River valley and crossed the divide to the Cantwell, mapping the topography and geology as far as the means would permit, and determining the position and altitude of Mount McKinley. Something was learned regarding the distribution of the placers in the Sushitna valley, and the source of the gold was traced to small quartz-seams in metamorphic slates.

Another expedition ascended the Skwentna from Cook inlet, portaged along the Alaskan range to the headwaters of the Kuskokwim, and following this great river to the sea, then turned eastward and, partly along the coast and partly inland, made its way back to the Pacific coast, completing in four months a journey of more than 1,400 miles.

On its way inland, the third party followed the then much-traveled Klondike route as far as the mouth of the White river, ascended that for about 100 miles, then crossed to the waters of the Tanana by portage, and followed that river to its mouth. The report of the topographic and geologic results of this expedition contained suggestions in regard to the distribution of placer-gold which have been verified by the recent discoveries in that field.

The other party, which was distinctly topographic, made its way inland along the same route and mapped some 2,000 sq. miles lying chiefly in the placer-district of the Forty-Mile region.

In that same year the U. S. Geological Survey was enabled, by the courtesy of the War Department, to assign two of its geologists to accompany army expeditions into the interior of Alaska. The one conducted explorations in the Copper River basin, while the other made its way inland from Cook inlet.

At the close of the season of 1898, the reports of these different reconnaissances and exploring expeditions not only increased the geographic knowledge of Alaska, but contained much practical information in regard to routes, trails and timber, as well as to the geology and mineral resources.

In 1899 the investigation was continued by two expeditions: One across the country from Pyramid harbor on Lynn canal to

Eagle City on the Yukon; and the other from the Yukon north to the Koyukuk. The first made its way along the northern front of the Saint Elias range, across the headwaters of the White to the Tanana, and thence, turning northward, reached the Yukon by a route through the Forty-Mile placer-district. On the way a hasty examination was made of the Porcupine placer-district and Forty-Mile region. What was still more important, it was definitely established that copper occurred in placers at the headwaters of the White and Tanana rivers; and the metal was traced to its bed-rock source.

The second party ascended the Chandlar, a northerly tributary of the Yukon, and reaching the basin of the Koyukuk by portage, followed that river to its mouth, giving special attention to the little-known placer-fields of the region.

Towards the close of the season of 1899, the two parties combined and utilized the few remaining weeks in visiting the newly-discovered gold-deposits of Nome, thus gaining material to publish a preliminary report on this important placer-field in the following winter. The wide circulation of this report instances the value of early publication of results.

The accompanying map on p. 383 shows in detail the progress of the topographic and geologic surveys of Alaska during the period 1898 to 1904, and gives also the distribution of the mineral products of that country as far as is known at present.

The increase in gold-output of the Seward peninsula, from $15,000 in 1898, to more than $2,000,000 in 1899, excited an interest which rivalled that shown in the Klondike. From the 20,000 people who went north during the summer of 1900, there arose an urgent demand for topographic surveys and information in regard to the mineral resources of the district. To meet these new developments, the purely exploratory work of the Survey was set aside for the time being, while activities were directed to areal mapping of the newly-discovered fields. About half of the Alaskan forces were concentrated in the Seward peninsula. These in the course of one season not only completed a contoured map of an area covering more than 6,000 sq. miles, but made a personal investigation of every placer-district and practically every creek of the southern half of the peninsula; as a result of which a report was issued the following year, showing the distribution of the gold-bearing

FIG. 1.—MAP OF ALASKA SHOWING MINERAL DEPOSITS AND PROGRESS OF TOPOGRAPHIC AND GEOLOGIC SURVEYS FROM 1898–1904.

gravels, the source of gold and its mode of occurrence. The question of bonanzas in the placers was the subject of special discussion. The theories advanced as to their probable origin have in most cases been confirmed by later developments in the region.

The discoveries in this season, of workable placers in the high bench, tundra and elevated beach-gravels of the Seward peninsula, furnished further proof of the value of geologic work; for the report of the previous year, in calling attention to these gravel-deposits then entirely unprospected, had dwelt upon the probability of their being auriferous. A similar instance was the discovery by a member of the Survey of stream-tin in the York region.

In the Copper River region there had been seemingly important discoveries of copper, and an urgent demand came from capitalists, mine owners and engineers, for more authentic data. To meet this demand a large force was dispatched to the new district and an area of about 5,000 sq. miles was mapped topographically on a scale of 4 miles to the inch. Within this belt a geologic reconnaissance map was made and the general occurrence of the copper-bearing rocks was determined, while individual prospects were examined as far as possible. The report which resulted from this work was of such a character that mining engineers had a definite basis for making preliminary estimates on the cost of railroad construction, as well as considerable data on the probable extension and value of the copper-deposits. During the same season, a more hasty examination was made of the copper-deposits of Prince William sound.

In 1901 the work in the Seward peninsula was extended to the northward by a reconnaissance-survey of about 5,000 sq. miles Some additional information was obtained regarding the distribution of stream-tin, and considerable areas were examined in sufficient detail to enable a fairly definite statement to be made regarding the presence or absence of placer-gold. A most valuable feature of the work, in the light of after-developments, was the outline of some granite intrusives; for it is along the contact of the sediments and granites that tin-deposits have been discovered.

The same year witnessed very important explorations in northern Alaska. One party, which had to travel 1,200 miles

by dog-teams in mid-winter, made a reconnaissance-survey from the Yukon northward to the Arctic ocean, traversing a mountain range which, up to that time, had never been visited by white men; and at the close of the season contrived a hasty examination of the Cape Lisburne coal-fields. Another survey, carried from the Yukon to Kotzebue sound, has proved of particular value to the prospector since the discovery of placer-gold within the area surveyed.

During this season, the Ketchikan mining district of southeastern Alaska also was subjected to a preliminary examination. Nearly 200 prospects were visited and reported upon, and a geologic reconnaissance-map was made of an area covering about 2,000 square miles.

To meet the growing demand for information in regard to the Copper River basin, two parties were sent into this district in 1902. These, besides completing the reconnaissance, topographic and geologic mapping of the basin, extended the work both to the north and south. A detailed study was made of the Chistochina gold-fields, as well as of the northern copper-field, which had received a preliminary examination in 1899.

The growing importance of the Yukon placer-fields and the consequent traffic on the Yukon river had led to a number of only partly successful attempts to mine coal for local use in that field. With the idea of furthering these interests, a party was dispatched to make a special study of the coal. The results showed that there were large areas of coal-bearing rocks on the Yukon, and that while the upper river coals were, for the most part, of lignitic character, some of those along the lower river, which are of a lower geologic horizon, can be graded as semi-bituminous, and should afford a local supply of fuel.

Hand in hand with these special geologic investigations, geographic mapping has proceeded by a series of reconnaissance-surveys. In 1902 an exploring party made its way from the head of Cook inlet through the Alaskan range, along its northwestern base to the Tanana river, and thence to the Yukon; completing an 800-mile journey in three months and a half. Yet topographic surveys were maintained throughout, and much was learned of the geology, as well as of the mineral resources. An area of coal outlined on one of the southern

forks of the Tanana, though of no immediate importance, has possible future value, for it lies near the route of a proposed railway from Cook inlet to the Yukon.

The demand in southeastern Alaska, where large capital has been invested in mining-plants, is for detailed geologic and topographic maps. But the great cost of such surveys unfortunately prohibits their extension with the rapidity required by the mining-developments. The beginning was made in 1902, by mapping an area of about 80 sq. miles in the vicinity of Juneau on a scale of one mile to the inch.

With improved transportation facilities, and better organization of the work, came a material decrease in cost, which enabled the U. S. Geological Survey to send seven parties in 1903. To one of these was assigned the task of completing the reconnaissance-mapping of the Seward peninsula, of which there are fairly accurate maps. At the same time an examination of the newly-discovered placer-fields in the northeastern part of the peninsula was made.

The general policy of the Survey is to keep in touch, as far as the appropriations will allow, with all mining-interests, and to publish from time to time supplementary reports based upon new data thus obtained. Thus, when the rapid developments in the older districts of the Seward peninsula created a demand for further geologic investigation, a party was sent to make a tour of nearly all the important mining-camps in the region. An interesting result of this work was the tracing of the tin to its bed-rock source. Though it is too soon to predict commercial importance for this new discovery, yet all indications are sufficiently favorable to stamp this as an excellent example of the importance of scientific investigation in a new mining-district, and a most striking case of the practical value of geologic work.

While every effort has been made to keep abreast with conditions in the more important mining-camps in Alaska, at the same time the lesser ones have not been neglected. The Yukon placer-fields have been steadily developing, though at a much slower pace than those in the Seward peninsula. To meet the great demand for accurate maps and information, two parties were sent into the Yukon country in 1903: One for a topographic survey extending from Eagle City to the Fairbanks dis-

trict, and thence northward to Circle City; and the other to subject the placer-fields of the Forty Mile and Birch Creek regions to a re-examination, and also make some preliminary studies in the newly discovered Fairbanks district.

The work of the previous year had thrown some light upon the occurrence of coal on the Yukon, but important stratigraphic problems still remained to be solved, which, while they had no immediate economic value, were still of sufficient practical importance to deserve a share in an investigation of the mineral resources. With this in view, another party was sent down the Yukon in 1903, which obtained still more evidence in regard to the age and distribution of coal.

Mention has been made of the detailed topographic survey of the region lying adjacent to Juneau. This district came in for geologic investigation in 1903, and the reconnaissance-mapping was continued over an additional area extending from Port Houghton to the Porcupine gold-district. The former received a detailed examination, and by careful study of its important mines the relations of the ore-bodies were determined and important conclusions reached in regard to their distribution.

While the interest in Alaska centers chiefly in its placer-mines, yet during the past three years considerable prospecting for petroleum has gone on. The successful boring of a well in the Controller Bay region and the favorable indications in the Alaskan peninsula led the U. S. Geological Survey to dispatch a party in 1903 to make a preliminary examination. Reconnaissance surveys in the Controller Bay region and in the oil-fields of Enochkin and Cold bays on the Alaskan peninsula yielded much data as to the geologic conditions of the distribution of petroleum. While these studies were only preliminary and cannot be considered conclusive, they at least show that these districts give promise of producing oil, and that drilling wells can be considered a legitimate operation. Furthermore, it was definitely ascertained that there are workable coal-seams of a semi-anthracitic character in the neighborhood of Controller bay. As this coal is of the highest grade yet found on the Pacific coast, it promises, in spite of the somewhat adverse conditions for shipping, to become an important resource of the territory.

In the foregoing an attempt has been made to outline briefly the character of the work which has been performed for the investigation of Alaska's mineral resources. While it is believed that the results are of practical value, a view which is borne out by the strong support that the U. S. Geological Survey has received from the mining men of Alaska, yet it is only too evident that much remains to be done, and that all of the investigations so far are only preliminary. It will be many years before exhaustive studies can be made, such as are now being carried on in many of the mining-districts of the United States.

In July, 1903, the Alaskan surveys, which had been at first a subordinate function of the geologic and topographic branches of the U. S. Geological Survey, were organized as a distinct division entitled, "The Division of Alaskan Mineral Resources," and its administration was vested in a Geologist in Charge. The force of workers, which in 1899 numbered only two geologists and two topographers, now includes ten geologists with two assistants, and three topographers with three assistants, besides an additional field force of about thirty men.

It is difficult at the present day to conceive how little was known of Alaska previous to 1898. The expeditions of earlier years had contributed something to the knowledge of the country drained by the Yukon, and more or less indefinite information was available in regard to a few other parts of Alaska, but the country as a whole was practically unknown. Many of the most important geographic features have been added to the map by the reconnaissance- and exploration-surveys of the years from 1898 to 1903, during which the principal mountain ranges have been outlined and the drainage areas defined.

A most important function of the Alaskan surveys has been the production of contoured maps, the value of which cannot be overestimated. There are demands for them from prospectors, engineers and capitalists, who find in them an essential aid toward the development of the country along any line. They indicate routes of travel to the prospector and explorer, railway- and wagon-routes to the locating engineer, and possible sources of water-supply to the mine-operator. During the six years that systematic work has been carried on, an area of between 90,000 and 100,000 sq. miles has been covered, representing about one-sixth of the total area of Alaska.

While the aim of the work has been to investigate mineral resources, this has, of course, necessitated geologic studies, for it is only through the medium of a thorough geologic knowledge that the facts gathered in regard to the value and distribution of ore-deposits can be practically and scientifically interpreted. From year to year this fundamental knowledge is growing, and in the course of time the geologic history of the territory will be deciphered. In fact, much is already known of the general succession and distribution of the many bed-rock formations.

The studies of the bed-rock geology contributed by each expedition are resulting in an accumulation of material which becomes increasingly valuable in tracing the distribution and origin of deposits having economic importance. The occurrences of such deposits, though a minor feature of the general geology, is the most important factor in the development of the country. The Survey's function is to study the relation of the two, and, in the course of these studies, results have often been achieved of immediate practical importance to the people of Alaska.

In the preceding paragraphs of this paper, reference has been made to the various classes of economic investigations which have been carried on in Alaska. This work is but fairly begun, for nearly all of the examinations have been of a most hasty and preliminary character. It has been the aim to establish general relations and to give the mining public the benefit of these by early publications of reports, leaving detailed studies to future years when developments will prove such as to warrant their cost. By pursuing this policy the Survey in the course of the past six years has been able to obtain some information from every mining-camp in Alaska, and this has, for the most part, been promptly published and widely disseminated. Recently Congress has manifested its interest in Alaskan mining-affairs by increasing the appropriation for investigation of the mineral resources to $80,000. This sum will make it possible to push the work at a pace more commensurate with its importance. The Survey's aim will be to keep abreast, if not in advance, of mining-developments.

The large investments made in water-supply ditches, pumping-plants and mining-machinery in the Seward peninsula has

shown the need of detailed surveys in this field. Though mine operators have many times been witnesses to the value of the present reconnaissance map, the magnitude of the mining-operations now necessitates a demand for detailed maps. Surveys for this will cost from five to ten times as much as the preliminary ones, but when completed will give the mine operator a thoroughly reliable basis for his engineering-work. While this will constitute the only immediate value of the contoured maps, yet they will prove of no less importance as a base for detailed geologic studies. These latter will not only determine the distribution of placer-gold, but will throw light on the much-mooted question as to the presence or absence of commercially valuable auriferous veins. There is a like demand for surveys in the Yukon placer-field, but here, developments have not yet advanced so far as to outstrip the usefulness of reconnaissance-maps. These should, however, be pushed to early completion; after which should follow detailed mapping of areas containing placers which have proved to be of high commercial value.

In the Cook Inlet placers the surveyor may see another field which demands attention. It has been the scene of such mining-activity as to warrant topographic and geologic surveys, but because of the urgency of other work it has, up to the present time, been almost neglected.

Owing to the great cost involved, the progress of detailed surveys in southeastern Alaska must of necessity be rather slow; but it will be possible within the course of a few seasons to complete the reconnaissance-mapping. This, in conjunction with a preliminary study of the occurrence and association of the ore-bearing horizons, should yield results of value to the prospectors.

The heavy capital which is being invested in the oil-fields on the Pacific coast of Alaska, single this out also as a region needing further attention. Hand in hand can go a study of the coal-fields of this district which give promise of large commercial importance.

The above suggestions do not by any means exhaust the possibilities for effective geologic and topographic work, but will indicate the lines along which there is the most pressing demand.

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TOPOGRAPHIC MAPS OF ALASKA.

The following maps are on sale at 5c. a copy, or $2 a hundred:

BARNARD, E. C. Forty Mile quadrangle; scale, 1:250,000.

PETERS, W. J. Juneau Special quadrangle; scale, 1:63,500.

The following maps are included as illustrations of published reports, but have not been issued separately. They can only be obtained by securing the report:

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Note on the Cost and Speed of Sinking the East Shaft of the New Kleinfontein Co., Benoni, South Africa.

BY EDWARD J. WAY, GENERAL MANAGER KLEINFONTEIN GROUP CENTRAL ADMINISTRATION, BENONI, TRANSVAAL, S. AFRICA.

(Baltimore Meeting, February, 1904.)

THE cost and the speed of sinking a shaft are factors of so great importance in operating a mine, that the data given in Table I., herewith published, will be of interest and value to all engaged in practical mining; and for this reason the data are communicated for permanent and accessible record in our *Transactions.*

The figures of cost cover the period from January 1, 1903, to May 31, 1903, during which time the total depth sunk was 858 ft.

So far as I know, the cost of sinking per foot (£13, 15s., 11.04d.) of a shaft of the dimensions 21 ft. by 6 ft. in hard rock, is the cheapest yet on record; and the speed of 213.5 ft. per month, which was attained during May, 1903, is also the best record for speed in sinking in rock so hard as that of this shaft. During the period from January 1 to May 31, 1903, there were 144 rounds worked, involving 4,032 holes, of a total depth of 28,963 ft. 2.4 in., which is equivalent to an average depth of 7 ft. 2.2 in. per hole bored. These data are of added interest from the fact that the work was done practically by native labor.

TABLE I.—*Cost of Sinking East Shaft, from January 1 to May 31, 1903.*

Labor, etc.	Cost. Total. £.	s.	d.	Per Foot Sunk. £.	s.	d.
White wages	3,292	16	0	3	16	9
Salaries	276	8	5	...	6	5
Native wages	1,190	4	8	1	7	9
Compound	616	11	3	...	14	5
Workshops	108	13	2	...	2	6½
Steaming station	1,353	3	3	1	11	6
Rock drills	2,620	0	2	3	1	1
Surveying	40	10	11	...	...	11¾
Transport	13	10	0	...	...	3¾
Traveling expenses	3	0	0	...	...	1

Stores. Description.	Quantity.	Value. £.	s.	d.			
Candles	207.2 boxes.	135	3	4	...	3	1
Detonators	86 boxes.	16	15	4	...	...	4¾
Fuse	2,475 coils.	42	3	6	...	1	.....
Gelatine	418 cases.	1,463	0	0	1	14	1½
Iron bars, etc	696 lb.	7	0	8	...	...	2
Steel bars	2,528 lb.	45	1	9	...	1	0¾
Rails	65,445 lb.	325	6	0	...	7	6½
Sleepers	259	18	1	7	...	...	5
Bolts and nuts	1,263	18	6	0	...	...	5
Dog spikes	363	7	2	4	...	...	2
Fish plates	3,011	35	19	4	...	...	10½
Nails, assorted	432 lb.	4	15	10	...	...	1½
Coach screws	82	1	6	8	...	...	¼
Oils, grease	.....	7	2	4	...	...	2
Piping	270 ft.	21	7	6	...	...	6
Tools	.....	11	6	8	...	...	5½
Timber, assorted	1,404 cu. ft.	71	16	0	...	1	8
Sundries	.....	89	5	9	2,321 0 7	...	2	1
.....	.....	.....			£11,836 19s. 5d.	£13	15s.	11d.

Total depth sunk, 858 ft. ; cost per ft., £13, 15s., 11.04d.

Testing Gold-Ores by Amalgamation.

BY ERNEST A. HERSAM, BERKELEY, CAL.*

(Atlantic City Meeting, February, 1904.)

INTRODUCTION.

THE small amalgamation-test of the laboratory is not always reliable as a basis for important decisions as to the character and commercial treatment of ores. The conditions of continuous practice on a large scale are not always reproduced in the brief laboratory-test. Sometimes such a reproduction would be very difficult, or impracticable; sometimes it is not even attempted, and the test is performed in a rough way, without regard to the effects of small variations in the methods pursued.

The aim of a laboratory amalgamation-test is to learn how much of the gold or silver in an ore can be economically recovered by amalgamation on a commercial scale. The yield can be increased, perhaps, by additional chemical treatment or by extreme comminution of the ore; but this might cost more than the value of such increase. Moreover, in practice, other operations often follow amalgamation; and these operations may remedy a defective extraction by amalgamation, more economically than could be done by directly increasing that extraction.

The test should be performed, therefore, upon a uniformly-sized material, and under conditions that are precisely known and capable of being exactly duplicated, or purposely modified.

The crucial feature of the test is the effectiveness of the contact made between the gold or silver of the ore and the amalgamating-surface. This may depend upon the temperature; the duration of contact; its force, due to gravity or impact; the hardness, plasticity or liquidity of the amalgam; the roughness or other quality of the amalgam-surface; the scouring action of the ore upon the amalgam, or the suspension and

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removal of metal once amalgamated; the supporting power of the currents; the fineness of the ore; the liberation of the metal from the gangue by comminution; the cleanness and freshness of the mineral; the molecular stress in the metal after grinding or rubbing; the oxidation and chemical changes of the ore during the test, and before the test; the kind of gangue; the kind of metal; the kind of amalgam; and the cleanness of everything. The test should be conducted with a realization of the significance of these conditions; and if so conducted, it often may give results capable of interpretation, and helpful as suggestions, though its general verdict be unfavorable.

Methods of Amalgamation.

Methods of laboratory-amalgamation are classified as follows, in accordance with similar commercial methods.

1. *Amalgamation During Grinding.*—This consists in wet crushing or grinding in contact with mercury or amalgam, usually in small arrastras, amalgamating-pans or other laboratory grinding-apparatus; followed, generally, by further addition of water, stirring, settling, and final collection of the mercury or amalgam containing the precious metal. Suitable chemical treatment often is applied to aid the process. Unquestionably this method amalgamates more of the gold and silver than any other. The material may be a crude ore, such as a "free-milling" gold-ore, or it may have been previously oxidized or chloridized by roasting. And the amalgamation may be either simple, or assisted by added chemicals, such as copper sulphate, sodium chloride, etc., often used with silver-ores—in which case the operation may be a complex chemical one. However conducted commercially (*e.g.*, in barrel, patio or arrastra), the chemical reactions generally require the aid of some accompanying positive mechanical or grinding-action when practised in the laboratory.

Grinding during amalgamation has been recommended also for hand-work with a compact field-outfit,[1] and has long been used by prospectors and assayers. For operations on a large scale, it has its economic counterparts in the arrastra-treatment of gold-ores, in stamp-milling with battery-plates, and in the

[1] R. W. Leonard, Assay of Auriferous Ores and Gravels by Amalgamation and the Blow-pipe, *Trans.*, xxv., 645.

numerous patented mills in which amalgamated plates are exposed to the pulp during crushing. In it, the freshly fractured mineral presents its surface of clean gold for amalgamation before the metal has been distorted by pressure, or coated by reaction with gangue, etc.

In a small test of this class, the amount of precious metal amalgamated generally is estimated from the difference between the assay-value of the ore, at the start, and that of the tailings at the end. The result, showing the percentage of free-milling gold in the ore, usually is higher than can be expected in any treatment on the large scale other than that of the arrastra. In such a laboratory-test, moreover, there is usually some difficulty in separating all the mercury from all the pulp. The quantity of pure mercury required is large, compared with the small amount of ore and gold present; the result when calculated upon differences in assay-value is less satisfactory than a positive determination (such as might be reached by retorting the mercury—which is an inconvenient and burdensome affair, preventing rapid, frequent and easy tests). Moreover, power is needed in the laboratory, or else much arduous and cautious hand-work must be performed; and, finally, there is uncertainty as to the exact degree and the uniformity of pulverization incidental to the treatment.

2. *Agitation with Mercury.*—This method consists in circulating, stirring or otherwise agitating a wet ore, already pulverized, mercury being added at the beginning of the operation, and, at the end, removed with the gold and silver it has collected. The results may conform to a large degree with those of ordinary stamp-milling and plate-amalgamation;[2] but here again, figures are obtained which, uncorrected, are likely to indicate possibilities not to be realized in practice.

It has been customary, always, to use a comparatively large quantity of mercury in these tests; and the retorting of so much mercury containing so little silver and gold is troublesome in accurate work. Therefore, in this, as in the grinding-test above described, the result generally is estimated from the difference between the assay-value of the original ore and that of the tailings. There is often difficulty, moreover, in causing

[2] H. A. Guess, *Journal of the Canadian Mining Institute*, 1898 (1), p. 10.

all the mercury to settle. Mercury, which has floured, may remain persistently with the metallic sulphides, or, when removed, may be mixed with sulphides carrying gold and silver that should not be reckoned as free-milling.

In the laboratory this method is used mostly for testing unroasted gold-ores, to compare with plate-amalgamation in milling. Such a use of liquid mercury has been found, by observing certain precautions, to give results corresponding to those of commercial practice with solid amalgam.

A roasted ore may require a positive grinding-action to break apart clots remaining from roasting. Such material, if tested by this method, must be thus mechanically prepared before the simple agitation with water and mercury begins.

In economic practice this test has its counterparts in the various patented devices for effecting an intimate contact between ores and liquid mercury. In the action of these devices, ore is made to ascend through liquid mercury, to flow across mercury or to fall upon mercury; it is impelled by gravity or centrifugal force; or it is moved by various agitators, aided by currents of water, and sometimes supplemented by heat, chemicals and electrolytic action. Much ingenuity has been devoted to such inventions; but they have not found general favor. As a rule, they do only the work of plate-amalgamation at a somewhat higher cost.

For a simple agitation-test, in which no crushing is to take place during the amalgamation, close attention must be given to the manner of preparatory crushing. By dry crushing, the gold contained in ore is rubbed, distorted and exposed to the mechanical action of every kind of impurity and gangue-substance present. Some of these substances may be harmful in the test, while they may or may not be equally so under the commercial conditions that the test is expected to imitate. The behavior of grease, graphite or talc is well known. Oxides, sulphides and other natural substances are also known to affect, chemically or physically, the refractory qualities of ore.

With wet crushing and immediate amalgamation, as contrasted here with dry crushing and subsequent wetting, there is partial elimination or segregation of some of these substances at the outset. Amalgamation can begin in practical, wet-crushing appliances often before all the ore is reduced to

the size to which it is destined to be crushed. Moreover, in wet crushing on a commercial scale, air does not reach the fresh fractures of ore, and can have no action in surface-tension; whereas, in dry crushing, films of air, or of dry solid material can remain upon the gold surfaces, and seriously interfere with amalgamation, especially if uncleanness has been permitted, or if the fines are not removed as fast as they have been produced.

In a simple treatment of this type, with any ordinary apparatus, an attempt to keep the mercury unbroken often gives too low a result. Treating the ore with water and only a moderate quantity of mercury, in cylindrical bottles, for example, by rotation around the axis, fails to draw the sulphides to the liquid mercury surface and thus expose all the material to amalgamation. Rotation in alternating directions gives the ore opportunity to settle during the change of direction, and causes the mercury to flow over the accumulated ore until finally the continuity of the liquid metal is broken. Attempts to economize to any great extent, in the proportion of mercury used, reduce the area of the amalgamating-surface until, in an extreme case, the mercury may be in the form of a globule, remaining at one end of the bottle, for a long time failing to come into contact with certain minerals of the ore, or with the gold which it is desired to amalgamate. A rotating spherical flask, or a specially constructed receptacle made by joining the ground tops of two glass funnels to retain the mercury in a V-shaped groove, is an improvement over the cylindrical flask; but even this permits the passage of certain constituents around the mercury; and complete amalgamation may fail. A cylindrical bottle, rotating on an inclined axis is subject to the same objection.

A grooved channel, more easily cleaned and managed than a closed receptacle, has been used in a few tests for amalgamating small portions of ore, before concentrating upon the wooden batea, the aim being to avoid excessive flouring of the mercury. This appliance was made by turning a deep annular V-shaped groove in the upper face of a wooden disk, 10 in. in diameter, of turned sugar-pine, shellacked and rubbed down to a dead finish. The disk is tilted at a constant angle by a pin projecting downward 1 in. from the bottom at the center. When rest-

ing upon a level surface, this pin gives the disk two points of bearing; the one being the lower end of the pin, the other, any point on the lower edge of the periphery. The ore is placed in the groove with mercury and water, and a slow, gyratory motion is given to the whole, by rolling upon a level surface. A small depression or shallow cavity is made at one point in the bottom of the groove to retain at least a part of the mercury, and to arrest particles of gold that travel more slowly around the groove than the flowing ore, water and mercury. A small quantity of mercury suffices for the test, and this can be recovered easily, with the gold that it contains, at the conclusion of the treatment. The test is subject to the objection that float-material, from certain ores, always forms when the surfaces upon which they lie are alternately flooded and drained. The device can be used sometimes, however, with the batea, and is convenient in its limited way.

Tumbling the ore with mercury and water, or rotating endwise in small bottles or tubes at a moderate speed, while the ore is in the form of a thin pulp, flours the mercury and fails to recover gold more expeditiously obtained by other means.

In all these agitation-tests, where the speed is moderate, the ore of a certain specific gravity, or gold of a certain degree of fineness, is liable not to make the necessary contact with the liquid metal. The mercury seeks the lower level, where it quietly flows, never making contact with certain parts of the ore.

3. *Agitation in Contact with Amalgamated Surfaces.*—By this method, the wet pulp, already crushed, is passed over amalgamated plates or agitated in contact with amalgamated surfaces, usually of silver or copper. This test, practiced with care, may yield results strictly comparable with outside-plate amalgamation. The surfaces employed may be in the form of amalgam-plates or of agitators of special construction, or they may be the amalgamated interior surfaces of retaining receptacles such as the pan, spoon or batea; but in any case they must be carefully prepared and kept in good condition. A copper surface coated with soft silver-amalgam is found highly satisfactory in such tests.

On a larger scale, the test can be made with a small stamp-battery and plates, such as some metallurgical laboratories possess, thus approaching still nearer, the exact conditions of

practice. Nevertheless, there are limitations to the practicability and scope of such tests; and it may often happen that the quicker and easier small test can be employed with advantage to settle some special questions. Many features of the large-scale laboratory-test have been brought out by Prof. R. H. Richards.[3]

Upon the smaller scale, this method has often been combined with a concentration-test performed at the same time, an amalgamated batea or other vanner being used for concentration. The clean-up and quantitative determination of the gold and silver from the comparatively large amalgamated surface necessary in such a vanning-test is, however, difficult; and generally the values amalgamated are estimated from a comparison of the assay-values of the pulp at the start and at the end. This difference-method for ascertaining the "free-milling" gold-value of an ore is objectionable. Moreover, in a small test with pan or batea, it is difficult to avoid some accidental loss of material; there is difficulty in preparing a satisfactory amalgamating-surface, and preserving it in good condition, while exposed to the air repeatedly during the test; and the small amount of ore that can be handled, and the irregularity of the treatment, make the results somewhat uncertain.

During a concentration-test on a wooden batea or miner's horn, one can recover much of the "free-milling" gold upon a fragment of soft, silver-amalgam or in a small globule of liquid mercury. Imperfect contact, however, between the gold of the ore and the amalgamating surface makes the result uncertain; and when mercury is used, and not amalgam, the breaking up and flouring of the mercury sometimes makes the method inconvenient. Nevertheless, it is possible to perform a kind of approximate, small-scale test in this way; and the direct determination for the "free-milling" metal is a satisfaction.

If the actual conditions of large-scale tests, with stamp-mill and plates, or other apparatus, are not present, there is little reason for operating large and inconvenient quantities of mate-

[3] The Development of the System of Amalgamation of Gold-Ores at Present in Use in the Mining Laboratory of the Massachusetts Institute of Technology, *Technology Quarterly*, February, 1890, p. 45.

rial. Between the large test on a commercial scale, and the small one of the assay-laboratory, the intermediate sizes have little to commend them. Large mechanical devices, circulating the pulp automatically over amalgamated surfaces are sometimes useful; but, in general, large size is an inconvenience, and the heavier machinery involved is not so closely held under inspection. Usually, a large plate-surface or a large quantity of amalgam or mercury is required; and this is more difficult to prepare and to recover with quantitative accuracy. To obtain the advantages of a systematic and positive small-scale laboratory-test for amalgamating a prepared pulp, the quantity taken should be as small as will permit good work in the original preparation of the sample.

The Size of the Test-Portion.

The size of a sample for amalgamation affects the details of manipulation and the significance of the test. There is a minimum quantity smaller than which a true sample cannot be; and this limit is controlled by the principles of sampling. Larger portions can be employed to an indefinite extent, but with progressively-increasing inconvenience and expense—and this without limit. It is the minimum limit only which demands consideration.

While the quantity of material used in a test generally should be as large as is demanded of an accurate sample, yet, with some ores and under some circumstances, even such a quantity may be unnecessarily and inconveniently large. Sometimes it is justifiable to work with smaller portions, provided the limitations of the sample are understood. The test is intended to show, accurately and positively, for the sample taken, the amalgamating-quality of the gold; but the average amalgamating-quality of a gold-bearing material may be represented truly by a sample so small as to be influenced in its assay-value by the possible occurrence, for example, of a nugget. Such an influence might vitiate a much larger portion, and must be guarded against by other means than by mere increase of the quantity tested.

For assay-purposes, visible particles of native gold are removed from the fine screens and are assayed separately, and these separate values are assigned to the larger portions from

which they were taken. In sampling for amalgamation, on the other hand, it may be desired to let this gold remain, in order to observe its action in the test. But such particles may vary from an infinitesimal size up to that of appreciable value; and thus may introduce an uncertain factor. It is known at the outset, that the sample may vary from a true average, by reason of the presence or absence of such particles; but if the total value of all the products of the test is known at the end, the original assay of a different sample is only of secondary importance; for the object of the test is to determine, not the average value of the ore from a given mine, but the proportion of gold that can be extracted by amalgamation, as represented in a given sample.

The possible size of these particles, that are allowed to remain, would ordinarily govern the necessary size of the sample. If we made unqualified provision for the possible occurrence of large gold particles, we should need to take a test-portion of almost hopeless magnitude. But when such particles are found in amalgamation they must receive special attention, and this, together with the method and manipulation employed, determines the necessary size of the actual test-portion employed.

To determine the relation between the coarseness of crushed ore and the necessary weight of the sample taken for the sole purpose of ascertaining average value, we can carry the relation between the weight and necessary fineness of an assay-ton sample to larger portions of larger mesh.

If one assay-ton be suitable for the assay when the fineness is 100-mesh, we may calculate how large a portion from coarser screens is required, on the supposition that the gold particles may be as large as the screen-opening, while all other conditions remain the same. In such a case, the weight of the sample ought to vary as the cube of the diameter of the closely-sized ore-particles which constitute it.

This principle may be expressed in the equation

$$W = zD^3,$$

in which D is the diameter of the ore-particles (gold, mineral or gangue); W, the weight of the sample needed; and z, a factor for each different kind of ore. If W = 1 A-T., and 0.0055

inches[4] be taken as the size of the opening of the 100-mesh screen, a numerical value for z is obtained, which, substituted in equations with other values of D, will give the values for W tabulated below.

TABLE I.—*The Necessary Weight of Sample, Where All Considerations Except that of the Coarseness of the Ore are Disregarded, in the Estimate of the Size of Sample Required.*

No. of Screen.	Opening.	Opening.	Sample.	Sample.	Sample.
	Inches.	mm.	Assay-Tons.	Pounds.	Tons.
100	0.0055	0.1397	1.000	0.0643	.....
80	0.0068	0.1728	1.890	0.1215	.....
60	0.0077	0.1957	2.744	0.1764	.....
.....	0.0098	(¼)	5.657	0.3638	.....
50	0.0100	0.2540	6.011	0.3865	.....
40	0.0120	0.3046	10.386	0.6678	.....
30	0.0173	0.4390	31.121	2.0110	.....
20	0.0270	0.6850	.....	7.6070	.....
.....	0.0394	(1.)	.....	23.6390	.....
10	0.0530	1.3450	.....	57.5400	.....
.....	(⅛)	3.1750	.....	754.8500	.....
.....	0.1576	(4.)	.....	1512.8970	.....
.....	(¼)	6.3500	.....	.....	3.019
.....	(½)	12.7000	.....	.....	24.155
.....	0.6299	(16.)	.....	.....	48.295
.....	(1.)	25.4000	.....	.....	193.241
.....	2.5196	(64.)	.....	.....	3090.920

The table shows merely a ratio between the coarseness of the ore and the corresponding weight of the necessary sample. It is recognized that this sample is of minimum size. Much more than this quantity would be desirable, were the duplication of assays impossible. With coarse ore, it is seen that the calculation leads to portions of enormous size.

The influence of a cube of gold of the full size of the screen-mesh indicated in the foregoing table would cause a variation of $1.05 in the reputed assay-value of the ore. The irregular shape of pieces of native gold might make this variation considerably greater. Two or three of these large gold-particles, or a gold-value that corresponds to them, will be found in excess or deficit, however thorough the mixing, or careful and uniform the sampling. A variation of several dollars per ton in the assay-value of the ore may be expected, therefore, from work with samples of native gold-ores of these sizes.

[4] The screen measurement here is that adopted by Brunton in his paper, The Theory and Practice of Ore Sampling, *Trans.*, XXV., 827 (1895).

Although the gold more frequently occurs in a finely-divided condition, yet large pieces of gold may occur in many ores, and are sure to be found in certain ores. Hence, unless we use very large test-portions, we must decide what size of gold particles can be classified as part of the ore, and what size must be removed or specially considered. Having done this, we can decide how large an ore-portion one must use in the test.

By making special allowance for small particles of gold in the sample, it is possible to perform rapid and sufficiently decisive preliminary tests with comparatively small portions of ore. The following statements are offered in justification of this opinion:

Amalgamation-tests are often made upon a series of sizes of the same material, to determine the degree of comminution most suitable for economically liberating the gold from the gangue. This series may include finely-crushed material, of which one or two assay-tons could be regarded as a fair sample. The large pieces of malleable gold, flattened and not crushed in obtaining such a finely-divided sample, must be removed from the ore to make the remaining small quantity uniform, and to protect the gold particles from further objectionable distortion in dry grinding. In the ordinary course of ore-dressing, and certainly in preparing an ore for amalgamation, this malleable gold, collected on the screens, is kept separate from the ore to be still further crushed. The value of the gold thus removed is assigned to the larger portion of ore from which it was taken; but its removal diminishes the assay-value of the finer ore passing the screens, so that the fine ore no longer represents the original value of the whole. The return of this coarse gold to the fine ore would make the sample necessary for assay or amalgamation as heavy as if the ore had not been thus crushed; and, consequently, this gold, once removed, is never again added; but a small sample of the finely-crushed and screened ore is assayed, and to the result the proportionate value of the coarse gold is added. But the laboratory-test should ascertain the behavior, in contact with the mercury, of this coarse gold, as well as the finely-divided gold which passed the screens. Hence, instead of putting this gold back into the finely-divided ore from which it was taken, and thus making a sample that cannot be fairly assayed or divided, it is better to

test by itself the behavior of the coarse gold, not unnecessarily obscured by the presence of inactive ore. In this way, the gold of a very large portion of ore may be represented in a small sample: The large pieces, by screening or picking them out from the sample; the smaller particles, by treating them in the finely-divided ore. All the information obtainable from an assay after exact sampling can be thus gained from a series of tests on a smaller scale.

Since all free-milling gold and its products can be recovered and assayed at the end of the small test, particles of gold, if large enough to require separate attention, influence the results in a way readily understood and interpreted. If they do not amalgamate, they are found unamalgamated in contact with the mercury. If they are absent from a given sample, but present in others, the fact is indicated in the final results, and can be explained by further tests. When large pieces of gold are absent in the whole ore-body from which the sample was taken, the small ore-portion is an accurate sample. Hence variation in the value of the samples need not obscure results relating to the possibilities of amalgamation.

The small portion permits a test more delicate and exact than can be conveniently made after increasing the scale. Sampling the several products becomes unnecessary; they are smelted in crucibles without subdivision, or, if small, are treated by scorification. The amalgamated gold and silver can be weighed and quantitatively determined, without rough handling or retorting.

A lot of ore received at the mill is never a true sample of the whole mine; and no single sample, however large, represents all the conditions that may hinder amalgamation. To obtain complete information it is often necessary to make tests upon ores and vein-material from different places, representing degrees of superficial oxidation, etc. As a rule, the various pertinent questions involved cannot be answered by a single test.

These considerations may often justify tests upon ore-samples as small as 100 grams; and such tests are frequently of great value. A sample of this size yields no more tailings than can be treated by a single crucible-assay, and contains sufficient value to permit the assay for gold and silver in any of the

products obtained. These products are scorified and cupelled if they are small, or fluxed in crucibles if they are large. A portion of this size can be regarded as a satisfactory sample of moderately fine ore. When the ore is coarse, it is always possible to make the tests in duplicate; but in that case, the gold itself, if also coarse, should receive special attention, as it would occur in few particles, materially altering the average value.

In tests upon 100-gram portions, each milligram of gold recovered represents $6.029, and each milligram of silver, 0.2916 oz. per ton. The 100-gram portion is also convenient in showing directly the percentage-weight of all the products obtained, and the values of these products per ton of original material are easily calculated.

The Laboratory-Test.

The chief purpose of the present paper is to describe a method by which the amalgamated gold is to be recovered, weighed and thus determined directly; the mercury or amalgam used is to be small in amount; the work is to be done automatically, as far as possible; and, at the end of the test, the pulp is to be left in a condition suitable for whatever concentration-test remains to be made upon the small portion of ore that has been used.

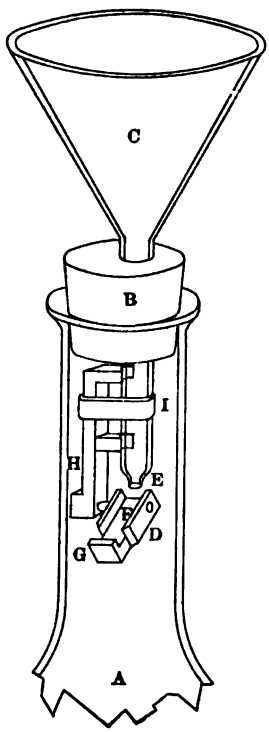
To secure an invariable certainty of the essential contact, with the smallest practicable quantity of mercury and an amount of ore relatively so large, requires the employment of a positive and uniform system. If liquid mercury be used, it must not "flour." Every particle of the ore must be brought into contact with an amalgam-surface that is small. The process must do more than effect, simply, an uncertain movement of a little mercury through a large quantity of ore; and no ordinary form of ore-conveyor can be reduced in dimensions and successfully applied to such small-scale work, by reason of the difficulty of cleaning, and of complications in small dimensions. The clean-up must be expeditious; the apparatus must be accessible and adjustable, and the action must be continuously open to the inspection of the operator.

Amalgamating-Funnel Apparatus.

Fig. 1 shows a simple appliance, made to test the practicability of some of these conditions, and found useful, repeatedly,

in the study of many details arising in connection with laboratory-amalgamation. The gold is collected upon a small amalgamated plate, placed in the carrier shown in the figure; and at the end of the operation this plate is removed, melted and finally parted. In this apparatus, other qualities are sacrificed to extreme simplicity; but its efficiency as a test-amalgamator must not be judged by its small plate-area. The possibility of observing the behavior of all parts of the ore with so small an amalgam-surface makes such an apparatus often most useful. In using it, however, strictly to determine the total proportion of free-milling gold contained in an ore, the ore must be passed over the plate more than once; and, for this purpose, modifications of the apparatus are more satisfactory; but in the small apparatus shown, a very high percentage of all the free-milling gold is collected in the first contact. Moreover, the operation can be repeated any number of times upon the same ore, without special attention, loss of material, or harm to product.

FIG. 1.



APPARATUS FOR PRELIMINARY TESTING AND OBSERVING THE AMALGAMATING ACTION UPON PULVERIZED GOLD-ORES.

The apparatus consists of a flask, A, provided with a rubber stopper and funnel, B and C, and a hard rubber attachment, D, for holding a small amalgamated plate. The operation requires that a uniform stream of ore, so small that at least half an hour shall be necessary for the passage of a 50-g. portion, should be delivered upon the amalgamated plate. Success depends upon skill in manipulation, the perfect condition of the amalgamated surface and the suitable size of the orifice, at which the flow of the ore-particles is restricted to a uniform stream, falling through water, which is ascending by displacement. Under the proper conditions, this flow of the pulp is uninterrupted, so long as ore remains in the funnel.

For the amalgamated plate, a suitable size has been found to be $\frac{1}{2}$ by $\frac{5}{8}$ inches. It should be as thin as practicable without danger of amalgamating through. Soft copper foil $\frac{1}{64}$ in. thick is sufficiently heavy. A silver-plated copper foil, or better, an amalgamated silver plate, is used when the assay has shown silver to be insignificant, and only gold is to be determined.

It is, of course, necessary that the whole surface of so small a plate be in perfect condition. Much of the gold amalgamates at the first point of contact, where the small stream falls upon the plate. A longer plate does not correspondingly increase the amalgamation. This first impact, at the point where the current changes its direction and spreads over the surface, should be carefully observed, as indicating much concerning the amalgamating-properties of the ore.

When silver foil is employed, it is annealed and cut into suitable pieces, which are amalgamated a short time before use. The surface may be roughened, if desired, by striking a sharp blow upon a fine-cut file placed upon it. It can be cleaned with dilute hydrochloric acid, and then, after washing in water, flooded with dilute nitric acid for a moment. After the removal of the mercury, the plate may be kept a short time, for the amalgamated surface to become somewhat hardened.

The carriage, D, is provided with a groove, F, $\frac{1}{16}$ in. deep, in which the plate fits nicely, and is held from sliding downward by the raised rim, G, at the foot. This rim is also important in guiding the pulp over the plate. Like the rim at the sides, it is $\frac{1}{16}$ in. high. Meeting this obstruction, the pulp parts into two streams flowing to the sides. Thus any fragment of coarse gold, that might be swept along in a straight line, reaches a turning-point near the foot of the plate where its high specific gravity takes it below the current of moving ore, and permits it to remain a longer time in contact with the mercury, and thus receive a better opportunity to amalgamate.

The carriage for holding the plate is attached to the support at the side by a small hard-rubber pin. (All metal should be avoided in the construction on account of its possible corrosion or amalgamation.) The plate thus can be inclined as desired, the angle depending to some extent upon the fineness of the ore, the shape of the crushed particles, the roughness of the amalgam-surface, and the specific conditions desired. But the

proper angle is higher than if water, flowing over the plate, helped to carry the pulp. There is here almost no current. The apparatus is filled with water at rest, the ore falls by gravity through it, each particle striking and traversing the plate. Hence an inclination of from 30° to 45,° according to the conditions of the test, is necessary.

Under the circumstances, this high angle is apparently not detrimental to amalgamation. On particles of a given size and specific gravity, the force of gravity alone may produce, at the higher angle, the same normal pressure of particle upon amalgam, as would the force of gravity combined with that of flowing water upon plates less highly inclined. On the other hand, in the treatment of particles of different specific gravity and size, the higher angle favors a more thorough amalgamating action upon such as would be easily suspended and carried away in flowing water.

The support, H, is held firmly to the stem of the funnel by the rubber band, I, indicated in the figure. Since funnels of different sizes may be used for different purposes, the connecting-pin is made sufficiently long to permit moving the carriage towards, or away from, the support, so that the amalgamating-plate may be always directly beneath the orifice E.

For most tests a 3-in. funnel is best. A portion of ore exceeding 50 grams would be desirable for numerous reasons; but the long time during which the ore remains stationary in the funnel makes it liable to become compact. With a 50-g. portion, however, this need not occur, if the orifice be of correct size and shape, and the ore fairly uniform in fineness. Towards the orifice, the stem of the funnel is drawn down abruptly to a smaller diameter, and the end is ground square, leaving a circular opening $\frac{1}{12}$ in. in diameter. An orifice as small as $\frac{1}{16}$ in. diameter may be used for fine ore, but the tendency to pack is correspondingly increased; and the slow passage of fine ore through the smaller aperature would make it necessary to reduce the size of the portion to something less than 50 grams.

Larger portions of ore, or portions that, for any reason, quickly become compact, can be held in suspension by directing a small blast of air through a capillary tube, from above to the top of the stem of the funnel. Ore once thoroughly wetted

will not retain air-bubbles in its downward course, so to interfere with the uniform flow through the orifice; but since the air-pump introduces a complication, and the whole of the ore is exposed to its oxidizing action for a large part of the time, and since, moreover, it causes a separation of the ore into heavy and light, and coarse and fine, which pass the plate in different times, the use of the air-blast is not generally desirable.

To make the test, the flask is filled with water, the amalgamated plate placed in its carriage, and the rubber stopper holding the funnel, support and plate is inserted. All air is thus excluded from the flask; and hence, when the stopper is inserted, a small excess of water is forced up into the funnel. This should be allowed to remain. The portion of ore to be tested is now added. This should have been carefully reduced to some one of the uniform sizes of ore usually adopted in such work, and have been screened to remove any foreign substance. It may be placed first in a small dish, moistened with water, and stirred about with the water to displace interstitial air, which otherwise might collect in a bubble at the orifice, and obstruct the stream by capillary action.

The moistened ore can be transferred to the funnel by the aid of a small stream of water from a wash-bottle, without loss of material and without the use of an excessive quantity of water. At once the ore begins to flow through the orifice in a uniform stream to the plate. Nothing need interrupt the continuous flow from the beginning to the end of the operation. Moving downward, over the surface of the plate, the ore causes a slight water-current after the start; and the plate, which might not seem sufficiently inclined at the first moment, immediately clears itself. This current can be started by a few grains of barren sand, when the conditions makes this refinement necessary.

After the ore has passed, if the operation is to be repeated, the water in the funnel is decanted into a second flask like the first. The second flask is then completely filled by further adding fresh water, and the funnel, support, plate and complete attachment is transferred from the first flask to this second one. The first flask, now containing ore that has passed over the plate once, and water that has been made turbid by the falling ore, is filled to the top with fresh water, to occupy the

space resulting from the removal of the stopper and attachment, and then inverted over the funnel, keeping the neck under water to hold the column.

A flask of the required neck-diameter cannot be inverted and inserted into a funnel of this small diameter without danger of losing some material, unless a cap or suitable cover is made to hold the column of ore and water while inverting it. A useful cap for flasks of this size can be made by turning a vulcanite disk $1\frac{5}{8}$ in. in diameter and $\frac{1}{8}$ in. thick, and cutting out the center concentrically to a depth of $\frac{3}{32}$ in. and a diameter of $1\frac{1}{2}$ in., leaving thus a raised rim, $\frac{3}{32}$ in. high, around the edge of a disk $\frac{1}{32}$ in. thick. A bent handle of glass or vulcanite may then be cemented to the disk and so shaped as to lie along the side of the neck of the flask when the loose-fitting cap is in place. The handle is held easily while placing the flask, and no trace of the contents ever is lost. The ore at once falls into the funnel; the slime settles; and the water remains in the flask. After sufficient time has elapsed for the ore and slime to settle from the inverted flask to the funnel, the flask is set aside, with its water unchanged, for a third operation, if such be desired.

The second passage of the ore over the amalgamated plate generally, but not always, yields a small additional quantity of recovered metal. With different ores this second yield averages less than 10 per cent. of the total free-milling value, but sometimes, by reason of defective first treatment, or otherwise, it is considerably more. A third passage over the plate generally shows that all the free-milling gold was collected by the first two. To learn how effective the previous treatments have been, one merely substitutes a fresh plate for the old one in a final test. By repeated treatment, the recovery of value thus quickly approaches a limit; and four repetitions can be regarded sufficient.

At the end of the test, the amalgam-plate is removed, washed with water, ignited gently in a muffle to remove the mercury, melted before the blow-pipe to produce a uniform alloy, and parted in nitric acid to recover the gold. When silver is to be determined, the thin copper plate is dried, after washing with water, scorified and cupelled as in the assay of an ore, or treated in a wet way, as some assayers prefer.

Larger portions of ore can be used only by placing a part of the ore-portion at a time upon the funnel. This increases the time required for the test; but throughout that time it may be seen that the plate is constantly performing its amalgamation-work as well as any corresponding area of surface could do, upon such an ore-portion as is put upon it.

The above method is useful in studying the action of different amalgams upon ore, in observing carefully the readiness with which the gold or silver particles of an ore amalgamate, and in obtaining from an assay-portion a preliminary idea of the free-milling qualities of an ore. The operation is simple; there need be no loss of material; and, at the end, the amalgamated gold and silver are quickly recovered.

Apparatus for Prolonged Amalgamation.

Fig. 2 shows a more effective automatic apparatus, adapted to systematic tests upon a somewhat larger scale. It causes all the ore to pass repeatedly over the small amalgam-surface, until the action is complete, or the desired conditions have been duplicated.

The apparatus, being made of glass, is fragile, and requires the use of a suction-pump, which introduces a complication. It is, moreover, restricted to certain small dimensions to produce the continuous automatic action desired. Opportunity is afforded, however, to observe the progress of amalgamation; and the simplicity of the process, when the apparatus once is made, the cleanness of all details and the thoroughness of the work, make the test fascinating to conduct and satisfactory in result.

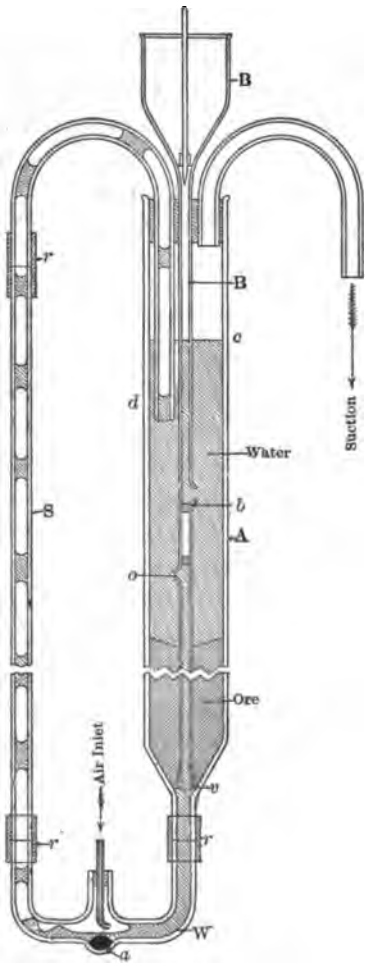
One form of apparatus upon this general principle was made in which the pulp was caused to fall upon and flow across an amalgamated plate like that shown in Fig. 1. The advantage of being able to replace the amalgamated plate with liquid mercury, however, is of much importance in a test of this character; and, as will be seen, means are provided to convey the pulp in actual contact across the level surface of the mercury without disturbing the liquid metal.

In Fig. 2, representing this apparatus, the amalgamating surface is shown as a globule of mercury, *a*. If liquid mercury is not wanted, a small amount of soft silver-amalgam, or a small amalgamated plate could be used.

The size of the apparatus can be increased to treat portions

of ore larger than those recommended here, but only within certain limits. The small tube through which the ore is elevated should have an inside diameter not much greater than $\frac{5}{32}$ in., to give the capillary conditions necessary to elevate all the ore with absolute certainty. The large tube, A, has been used with a diameter of 1 in. and a length of 23 in. above the valve, *v*.

FIG. 2.



AUTOMATIC APPARATUS FOR PROLONGED AMALGAMATION TREATMENT OF GOLD-ORE SAMPLES.

The tube, A, is securely clamped to a suitable support. The funnel tube, *B*, is left free to slide up or down through the stopper at the top of the tube, *A*, thus permitting the opening or closing of the ore-passage, *v*, at the bottom. The main apparatus is thus held rigidly and permanently in position, the only detachable parts being the lower tube, *W*, containing the amalgam, *a*, and held in place by the rubber connectors, *r*, *r*. The air-inlet should be a substantial capillary tube drawn down to a diameter of $\frac{1}{75}$ in. at the orifice at the bottom, and turned at its lower end to form a bend, as shown in the figure, in order to cause the current of air to impinge upon and against the flowing stream of water and ore.

A small blow-hole, 0.01 in. or thereabouts in diameter, can be made in the side of the elevator tube above the rubber connector, *r*. This aperture can be kept closed, except when needed, by covering it with a short piece of rubber tubing like the connector, *r*, fitting

around the tube and sliding over the opening to close it. Such an opening insures a lighter column of material in the tube, *S*, than that in the tube, *A*, and is a safeguard against the reversal of direction of the circulating material. In ordinary working, however, it is unnecessary; and the aperture should remain usually closed.

At the beginning of the operation, the ore-tube, *A*, being supported firmly in a vertical position, the tube, W, is removed from the rubber connectors, *r* and *r*, and the mercury, or amalgam, *a*, is placed in the depression at the bottom of the W-tube. A globule of mercury, 0.5 gram in weight, will suffice for the test, or a small, flattened particle of soft silver-amalgam can be used. The capillary air-inlet is inserted, but at the beginning it is drawn upward somewhat from the position in which it is seen in the figure, and is turned in the opposite direction, so that the orifice points in the direction of the moving current, and not against it, as is shown in the figure. The tube, W, is then attached by the rubber connectors, *r* and *r*. The funnel-tube, B, is now pushed downward, closing the valve at *v*. This valve, *v*, can be made by grinding the stem of the tube, B, into the contraction of the tube, A, until it becomes a closely-fitting glass valve; or better, the lower end of the tube can be covered with a short piece of thin, tightly-fitting rubber tubing, which will conform to the shape of the valve-seat formed by the contraction of the tube, A, if left unground. The severe usage a valve in this position is likely to receive makes the latter course satisfactory, as well as simple in construction. This valve should remain closed while the ore is being added.

In adding the ore, it is preferable to moisten it first, so as to avoid capillary difficulties due to interstitial air. A convenient portion is 100 grams, which will occupy a height of 10 or 12 in. in the tube, A, leaving sufficient space above for water; the ore never rising as far as the orifice, *o*. The coarseness may range between 20- and 120-mesh, according to the purpose of the test.

The ore is poured into the apparatus through the funnel-tube, B, the stopper in the neck being removed, and the passage of the ore accelerated, if necessary, by attaching the suction-tube to the pump. The last part of the ore is washed in with a fine

jet of water from a wash-bottle, thus avoiding an excessive quantity of water. The ore passes downward through the stem, B, until it meets the obstruction, *b*, and passes into the outer tube, A.

The ore having settled, the suction-pump is running uniformly, and the tube, A, is partly filled with water, above the ore. Air is passing through the capillary air-inlet, up through the tube, S, and out at the "suction"-tube.

Sufficient water is now added to the apparatus to raise the level to *c*. The tube, S, should dip beneath the surface of this water (as shown at *d*) in such a way as to cause some agitation of the water from the air entering through S, and thus give the ore entering the tube, A, from the tube, S, opportunity to fall through the column of water, and not to be blown and spattered upon the sides of the tube, A. This water is added, as was the ore, by lifting the stopper in the tube, B.

While the apparatus is being filled with water, there will be a current of water passing through the stem, B, into A, back through the aperture, *o*, into B, and thence downward through *v*, where no obstruction to the flowing water is met, while the ore is held back in the outer tube, A, by the continued closure of the valve, *v*. The water thus passes downward through the orifice, *v*, into the tube, W, in which the air, entering through the air-inlet, meets the moving current of water and impels it onward, while the orifice is turned in the direction of motion. The water, now mixed with air, passes into the side-tube, S, and rises, on account of the difference in head between the solid column of water in A, and the air-and-water column in S. The water is thus passed repeatedly around the apparatus and over the mercury surface—but thus far no ore has been permitted to pass over the mercury surface, and the exact adjustment has not been made which produces the correct velocity of the current over the mercury.

At this time the air-inlet may be turned so as to direct the jet of air against the stream and to cause a depression of the surface of the water in the tube, W, directly over the mercury. The condition of the water-current at this point is under perfect adjustment. It is desired to decrease the cross-sectional area of the stream directly at the point where the mercury is placed, until the correspondingly increased velocity will keep

the mercury surface clear when the ore is passing. This adjustment is made by raising, or lowering, the air-inlet tube—a minor regulation, the extent of which is governed by the size of the capillary orifice and the suction-pressure.

The air itself must not come in contact with the mercury, and too shallow a stream of pulp over the mercury surface would cause the mercury globule to break and be swept along with the pulp, thus injuring the test in one of its most important features. It is possible to thrust the air-inlet tube downward so carelessly and so roughly as to drive all the mercury up through S, and over into the ore; but this never need happen. There is no difficulty in keeping the entire portion of mercury in one mass, unbroken, after once knowing the action of the air- and water-currents in the apparatus.

So far it has been assumed that the apparatus has been working upon water alone, and that ore has not been permitted to pass the valve, *v*. Fig. 2 shows the apparatus in this state. The ore now can be admitted by raising the tube, B, a short distance, when the ore passes downward through the valve, *v*, and meets the downflowing water which is also passing downward through *v*. There must be a free passage of ore on all sides of the stem of the tube, B, at the valve, *v*, that no part of the ore may be held back from circulation. The ore and water mix directly below *v*, and the adjustment of the valve regulates the proportion of ore to water in the pulp.

The thin pulp now passes through the tube, W, where its behavior at the amalgamating-surface is under perfect adjustment by the regulation of the capillary air-inlet tube. If the ore-particles tend to lodge around the mercury more than is desired, the inlet-tube can be pushed downward somewhat. Never, however, must the air be directed in such a way as to agitate the surface of the stream above the amalgam and break the current into bubbles directly over the mercury. The orifice of the air-inlet tube is lowered to such a level that the impinging current of air directed against the stream, without breaking into bubbles, glides over the surface of the water against the current. The air-current then turns back upon itself, follows the water back past the orifice whence it issued, and finally breaks into bubbles in the upper part of the limb of the W tube, as shown in the figure.

The pulp passes up through the tube, S, without lodgment of any of the heaviest ore-particles and thence over into the tube, A. The operation is continuous. Every particle is drawn across the amalgamating-surface as many times as is desired, and the behavior of the metal upon the mercury can be easily seen in this tube of small diameter. The conditions are favorable for amalgamation at each passage of the ore, for there is perfect control of the two opposing-tendencies; the one, the tendency to lodge, caused by insufficient velocity of current; the other, the tendency to clear the plate or disturb the mercury, caused by high velocity. This is regulated in either case by the position of the capillary air-inlet tube.

The suction-pump, by means of which the circulation is effected, must be uniform in action. Any pump that produces a constant suction without pulsation can be used. The equipment of almost every laboratory, nowadays, demands a supply of tap-water under a fairly constant head; and when such water-supply is available, the Richards pump[5] is found most serviceable. While the ore, ascending through the tube, S, appears to be exposed to conditions favoring oxidation, this action is but momentary, and, as is usual in amalgamation, the ore is exposed to the action of aërated water.

At the close of the amalgamation-test, when it is desired to stop working and effect a separation of ore from mercury, the tube, B, is simply lowered and the valve, *v*, thus closed. The apparatus now begins to work upon water alone, ore no longer being admitted, and the mercury surface washes free from adhering ore by the water-current. The capillary air-inlet tube is now drawn upward somewhat, and turned carefully to deliver its air in the direction of the current, in such a way as to avoid disturbing the mercury.

The operation is continued after closing the valve, *v*, until all the ore-particles have settled to the bottom of A. The W tube next can be drawn downward some distance through the rubber connectors, *r* and *r*, which fasten W to S and A; but before removing W entirely, the connector at the right is closed with some form of pinch-cock, to avoid a rush of water through W when the connection is first broken. The suction

[5] *Trans.*, vi., 492.

is then stopped, and W is taken away from the apparatus entirely, leaving the rubber connectors behind—the one at the right closed with the pinch-cock retaining all the water in A. The contents of W can be removed without inconvenience or loss, and the mercury can be evaporated, scorified or parted, as desired, in order to obtain the gold and silver.

The ore, A, is now allowed to flow out into a suitable dish, by opening the valve, *v*, and removing the pinch-cock attached temporarily to the rubber connector. The material so collected is ready to be treated in any other test desired, or to be dried and assayed.

In any ore crushed and treated without sizing, there is always an appreciable amount of slime. This slime remains persistently suspended in the water under any treatment, and is to be recovered at the end, and its importance noted. Only a small quantity of water being required for the test, the slimes will be obtained in a somewhat concentrated form. All this water, with its suspended material, is saved; the solid material is collected from it, by settling or filtration, and the sediment is dried, weighed and subsequently assayed, or treated otherwise, according to the needs of the case.

At the conclusion of the test with ordinary gold-ores, the value is in three portions: Free-milling gold, gold in the tailings, and gold in the slimes. The last would be lost in certain processes of further treatment; and the extent of the loss can be determined by drying and scorification-assay of the slimes. Often, on the other hand, it is desirable to make only one portion out of slimes and tailings.

Ordinarily, the gold of the tailings, whether including the slimes or not, should be treated by some concentration-process to complete the test. This subsequent test can be made with the batea, horn-spoon, pan, hand-jig, or other suitable device.

Finally, after concentration, the value is in four portions: Free-milling gold, concentrates, tailings and slimes. Except the free-milling gold portion, which is to receive other treatment, these should be dried and weighed, and the percentage of concentrates, tailings and slimes should be calculated from these weights. The whole of each portion then should be used for the assay. From these assays are calculated the values in dollars of gold and ounces of silver upon the double basis:—

(1) The value in gold and silver of each ton of each product obtained in this manner. (2) The value in gold and silver that one ton of the original material would yield in the form of the products, (a) as free-milling gold, (b) as concentrates, (c) as tailings, (d) as slimes. The sum of these values shown by this second calculation is the assay-value of the original ore, and should check with the results of the regular assays of the ore, as far as sampling will permit.

The duration of the operation must depend upon the rate of flow and the size of the sample used. The first passage of the ore over the mercury amalgamates a large part of the free-milling gold, the exact proportion varying in different ores. In a treatment of 100 grams for two hours in this apparatus, the ore flows across the mercury about ten times; and this time may be regarded as long enough for results comparable with economic milling. A trifling difference in the rate of flow, which may decrease or increase the number of passages of the ore, is compensated by the amalgamation-efficiency of each passage of the ore, so that a treatment of this duration, with this approximate rate of flow, can be regarded as doing work of a definite character. The rate of flow should be regulated according to the behavior of the pulp as it passes over the mercury surface, to make sure that every particle is drawn over that surface in actual contact.

Nothing prevents the use of larger samples of ore but the longer time required and the need of dividing the resulting products for sampling and assaying. The advantages of the method are brought out more fully by employing portions not too large, and by an apparatus of the moderate size here described. The apparatus, once in place, is found ever ready for use, and the time and attention required to use it are but trifling. The small quantity of amalgam is recovered without loss or inconvenience, and the value of the metal amalgamated is learned in a short time by evaporating the mercury, fusing the metallic residue before the blow-pipe, parting and weighing; or, if the case demands it, by scorifying, cupelling and parting, as in ordinary assaying.

General Considerations.

Any test is found to possess most merits and fewest limitations when reduced to the smallest possible dimensions permit-

ROBERT HENRY THURSTON.

ting uniform action and certain treatment. A limit is always reached at some point in this reduction in size, where some condition no longer can be fulfilled. In many continuous mechanical tests the limit is set by irregularity, resulting from complexity and intricacy. In tests of a complicated chemical nature, the instability of small masses and fluctuations in physical surroundings make the work often difficult to practice.

The amalgamation-test, however, meets no actual barrier to reduction of scale, until the difficulty of procuring a sample is encountered. Down to this point the obstacles are eliminated in the method described. A further decrease in the size of the portion would necessitate a finer pulverization of the sample; and this is prohibited in the purpose of the test. By giving heed to large pieces of gold when they occur, this necessary portion is reduced to a minimum size; and under these conditions the test is found often valuable.

Biographical Notice of Robert Henry Thurston.

BY R. W. RAYMOND, NEW YORK CITY.

(Atlantic City Meeting, February, 1904.)

In the death of Professor Thurston, every branch of American engineering has suffered a great loss; and although his career has been more fully described in the publications of societies and institutions with which he was more closely connected by virtue of his special distinction as a mechanical engineer, some tribute to his achievements and his character is certainly due from this Institute, in acknowledgment both of his professional eminence and of its own peculiar obligations to him. In addition to this self-evident reason for the present notice, I offer in explanation of my own act in claiming the privilege of preparing it, instead of resigning that task to some more competent hand, the argument of a long and unbroken personal friendship.

Robert Henry Thurston was born October 25, 1839, at Providence, R. I., where his father, Robert L. Thurston, was then a manufacturer of steam-engines; and in frequenting, as a youth,

his father's establishment, he developed what was, no doubt, an inherited genius and taste for mechanical engineering.

Whatever may be generally the best order in the stages of technical education—whether shop-training should precede, accompany or follow school-study—I think that, to mechanical engineers, at least, actual familiarity with machinery is an invaluable preparation for the comprehensive study of theory.

Certainly, young Thurston was no exception to this rule. He added to his preliminary training a course in the scientific department of Brown University, graduating in 1859, at the age of twenty, as Bachelor of Philosophy, and returning as consulting and designing engineer to his father's business.

In 1861, he entered the U. S. Navy as an assistant engineer, and served until the close of the War of the Rebellion, being attached to the fleets of Dupont and Dahlgren, and, in 1864, commissioned as First Assistant Engineer in the Steam-Engineering Corps. At the close of the war, he was sent as Acting Assistant Professor of Natural Philosophy to the U. S. Naval Academy at Annapolis, where he remained until June, 1871. In that year the Stevens Institute of Technology was opened at Hoboken, N. J., and, through the wise judgment of Henry Morton, its distinguished first President, Thurston was called to the chair of mechanical engineering in that institution, the reputation of which, as well as his own reputation, was greatly advanced during the fourteen years of his professorship by his work as an instructor, investigator and contributor to technical literature.

In 1885, he became Director of the new Sibley College of Engineering at Cornell University, Ithaca, N. Y., a position which he retained to the end. His work in this wider sphere comprised the progressive organization and development, as well as the continuous administration, of the courses of instruction and the acquisition of the new buildings, plant, apparatus and corps of instructors required by the growth of the college in numbers and in fame. For, although the grants and gifts necessary to this result came from others, it was unquestionably his inspiring enthusiasm and executive skill that secured them. The story may be stated in a few significant figures. In 1886, at the first commencement after Professor Thurston's appointment, Sibley College had 8 graduates, and a total

of about 60 undergraduates. In 1903, there were 120 graduates and nearly 1,000 undergraduates.

Meanwhile, his literary activity was intense and continuous. I have never seen, and cannot undertake to compile, a full list of his books and professional monographs. They are said to number more than 300. Among the more important books may be named the "History of the Steam-Engine" (1878); the "Materials of Engineering" (1882–6); the "Manual of Steam-Boilers" (1890); "Engine and Boiler Trials" (1890); and the "Manual of the Steam-Engine" (1890–1).

I quote from the N. Y. *Engineering News* of Nov. 5, 1903, the following summary of his work as an investigator and inventor:

"Besides his other activities, Dr. Thurston acquired some reputation as an inventor, chiefly in connection with his work in original investigations. His oil-testing machines and his torsion-testing machines are most familiar to engineers; but he also made inventions in magnesium-burning lamps, signal apparatus for the army and navy, various machines for testing the properties of iron and other metals and for ascertaining the properties of lubricants, and in addition to this work he made improvements in steam-engines and in existing scientific apparatus. He performed much original work in scientific investigation in connection with engineering problems, and he is credited with having organized in 1872–73 the first laboratory in the United States for research in the applied sciences of engineering. Among some of the more important results of this work of Dr. Thurston may be mentioned the determination of the useful qualities of alloys of copper and tin and copper and zinc; his studies and conclusions upon boiler explosions; his investigations of the laws of friction and lubrication, and of the laws of variation of engine-waste in heat and power; and his studies in the field of the commercial economy of steam-engines.

"As an engineering expert, the United States Government frequently demanded the services of Dr. Thurston. He was a member of the U. S. Commissions sent to the Vienna International Exposition of 1873 and to the Paris Exposition of 1889; the U. S. Commission to investigate the cause of boiler explosions (1875); the U. S. Commission to test iron, steel and other metals (1875–1888); the U. S. Commission on Safe and Bank-Vault Construction (1891), and the U. S. Board to report on the construction of the iron-clad "Puritan." Dr. Thurston became a member of the American Society of Civil Engineers on December 6, 1871; he was the first President of the American Society of Mechanical Engineers, holding that office two years, and he was three times Vice-President of the American Association for the Advancement of Science. He was also a member of several European scientific associations, and he had received the degrees of C.E., Ph.B., A.M., LL.D. and Dr. Eng."

Professor Thurston joined the American Institute of Mining Engineers in 1875, at a critical period in its history. There was still room to doubt whether the Institute, then only four

years old, would succeed in establishing itself as a national society. Mining engineering and metallurgy involve so much of civil, mechanical and electrical engineering on the one hand, and of chemistry and geology on the other, that the existence, or the inevitable future formation, of separate societies, representing these branches, might easily hinder the growth of the Institute. At this juncture, the cordial co-operation of such distinguished civil and mechanical engineers as Holley and Thurston turned the scale. Professor Thurston's contributions to the *Transactions* comprise the following:

	Vol.	Page.	Year.
New Determination of the Coëfficients of Friction of Lubricated Journals, and on the Laws Governing Such Friction, ...	VII.	121	1878
Remarks on the Torsional Testing-Machine, ...	VII.	199, 201, 366	1879
Remarks on the Work of the U. S. Test Board on Bronzes, . . .	VII.	263	1879
Remarks on Steam-Boilers, . . .	XIII.	722	1885
Remarks on the Use of the Triaxial Diagram, etc., ...	XXVIII.	894	1898

The latest of these contributions refers to the exceedingly ingenious and original device which Professor Thurston had employed in 1877, in his report on copper-alloys to the U. S. Testing Board, for the graphic representation of the relation between the composition and the physical qualities of ternary compounds.

With the exception of this interesting communication, it will be seen that his contributions to the *Transactions* ceased in 1879. This was the natural result of the organization, in 1880, of the American Society of Mechanical Engineers, and the election of Professor Thurston as its first President. But though his professional contributions were thereafter mainly made to the new society, he never relinquished his membership, or lost his friendly interest, in this Institute. In a word, by reason of the loyal friendship of Holley and Thurston, and many others like them, this Institute has been strengthened and helped by the creation of younger societies, to which it may be said, in some sense, to have given birth, and with which it has always maintained fraternal relations. Nobody knows

better than I how easily a different result might have been brought about; and therefore nobody has a better right to declare how much this Institute owes to such men as Thurston.

If I were required to specify Professor Thurston's greatest power and source of influence, I should say it was spiritual, even more than intellectual. At least, there was a spiritual element crowning all the rest, without which all the rest could not have carried him so far. The magnetism of his mere presence; the enthusiasm of his sympathetic interest; his infectious joy in the pursuit of knowledge,—these things created about him an atmosphere of inspiration and stimulating encouragement. As a boy in a public school, I once had a teacher who possessed this gift, and was able, by reason of it, to teach with wonderful success even sciences and languages of which he knew little or nothing more than his pupils. He and they studied such subjects together at recess and after school-hours, with a fierce determination and delight, of which he set the example. If that country schoolmaster could have had also what his obscure position had denied him,—namely, a comprehensive knowledge, as well as an ever-fresh thirst for knowledge,—and if, while conscious of high attainment, he had still retained the energy of still higher ambition, and the compelling power of leadership, he would have been something like what my acquaintance with Robert H. Thurston leads me to believe him to have been. I need hardly add that such buoyant hope, courage and joy of work as Thurston always seemed to express, cannot be thus imparted until they have first wrought their effect upon their possessor. Lazy men do not rouse others to activity; cowards do not gloriously lead columns to victory. All Professor Thurston's life was behind his cordial, stimulating presence. It must have been a great privilege to have him for a teacher. I know it was a privilege to have him as a friend.

His death on October 25, 1903 (the sixty-fourth anniversary of his birth), was sudden and peaceful,—without any recognized preliminary warning, though not without some premonitions of indisposition, the significance of which was not understood until afterwards. He was, intellectually, still in his prime, and might have done much more good work if he had lived longer. But he had accomplished already a strong man's

task, and was entitled to reward and rest. Whatever dread, insoluble mysteries may be presented to us in earthly lives wrecked by error or misfortune, or prematurely cut short by death, this life raises no such questions. Fruitful, complete, blameless and happy, it furnishes only a theme for gratitude, and an example for imitation.

Professor Thurston was married in 1865 to Susan Taylor Gladding, of Providence, R. I., who died in 1878. In 1880, he was married to Leonora Broughton, of New York City, who, with three daughters, survives him.

Biographical Notice of William Henry Pettee.

BY R. W. RAYMOND, NEW YORK CITY.

(Lake Superior Meeting, September, 1904.)

By the death of Prof. Pettee the Institute has lost one of its earliest, most distinguished, most useful and most beloved members. Among those who survive him there are few who have rendered services of equal professional value or unselfish devotion. The warrant for this statement will be amply furnished in the present brief notice of his life and work.

William Henry Pettee was born, January 13, 1838, at Newton Upper Falls, Mass., where his childhood and youth were spent. His father, Otis Pettee, a manufacturer of cotton cloth and of cotton-mill machinery, died in 1853. Up to that time, the son had been educated mainly in the district school; but, being inclined by temperament and constitution rather to books and study than to physical exertion, he had been wisely restrained in mental activity, and required, for a season, to work regularly in the foundry and machine-shop of his father. Unquestionably, this discipline not only benefited his health, but also constituted a valuable preparation for his subsequent professional career.

In 1857, after suitable preliminary study at various New England academies, he entered Harvard College, where he took high rank in the classical course then required, was selected in his Junior year to deliver a Latin oration; was elected a mem-

ber of the Phi Beta Kappa Society (an honor well known to be the recognition of a high record for scholarship); and was graduated with distinction in 1861. The benefit of this classical training was clearly evident in his later life. It may not be possible for all students to preface or accompany their technical education with a course giving general culture and command of language, and certainly no such general course can take the place of thorough technical study; but there is no room to doubt that (other things, such as genius, health and opportunity, being equal) the fortunate few who possess both, easily go to the front.

In his Junior year, Pettee had taken, as elective studies, Spanish (under James Russell Lowell) and chemistry. After his graduation, he pursued for three years post-graduate studies in the Engineering Department of the Lawrence Scientific School and in the University, acting also, for part of that period, as assistant to Prof. J. P. Cooke, in the teaching of chemistry.

In August, 1865, he went to Europe, where he spent three years, partly in visits to mining-districts, but chiefly at the Royal Mining Academy at Freiberg, Saxony. In the autumn of 1868, he returned, to become an instructor in the School of Mining and Practical Geology, established in that year at Cambridge, Mass., by Prof. J. D. Whitney. In 1869, he was formally appointed Instructor in Mining, and, in 1871, Assistant Professor of Mining.

In the summer vacation of 1869, Prof. Pettee made, under the auspices of the Cambridge School of Mining and the direction of Prof. Whitney, a geological and topographical reconnoissance of the South Park, in Colorado—a region which had not been covered, at that time, by the work of the U. S. Geological Survey.

During the college year of 1870–71, Prof. Pettee received a leave of absence, permitting him to assist Prof. Whitney in the State Geological Survey of California. To this work he devoted fourteen months, mostly spent in the study of the auriferous gravels of that State, and of problems connected with the barometric determination of altitudes. Some of the results of these studies were published in 1874, by authority of the California legislature, under the title "Contributions to Barometric Hypsometry," to which a supplementary chapter was

added in 1878. This report embraced the collation and discussion of a vast amount of material, laboriously gathered from scattered sources, foreign as well as American; and it has taken its place in technical literature as one of the (not too frequent) instances of work so well done that it need not be done again, but may be built upon as a safe foundation by later investigators. Prof. Whitney's high estimate of the labor, learning and critical judgment with which it was performed by Prof. Pettee, was expressed in a prefatory note to the volume issued in 1874.

Returning to Cambridge as Assistant Professor of Mining, Prof. Pettee continued for four years, under that title, to give instruction in physical geography, geology, meteorology and other branches. But the original scheme of the School of Mining and Practical Geology was not carried out.

Meanwhile, instruction in some branches of mining engineering, and particularly in assaying and metallurgy, had been given at the University of Michigan prior to 1875, and in the years from 1868 to 1871 the degree of M.E. had been conferred upon a few candidates. But in 1875 the Legislature of Michigan provided for the formal establishment of a School of Mines at the University; and in the autumn of that year, Prof. Pettee, having accepted the chair of mining engineering in the new school, entered upon his duties at Ann Arbor, Mich. For nearly 29 years he continued his connection with the University, of three of the faculties of which he was a member, besides serving, by annual re-election for many years, as the Secretary of the University Senate, and editing the annual reports and other publications of the University.

In 1879, he received a leave of absence, to continue his study of the auriferous gravels of California. His report on this work was published by the Museum of Comparative Geology at Cambridge, Mass., as an appendix to the first volume of Prof. Whitney's *Contributions to American Geology*.

Prof. Pettee was one of the original Fellows of the Geological Society of America; a Fellow of the American Association for the Advancement of Science (of which he was elected General Secretary in 1887); a member (during his residence in Massachusetts) of the American Academy of Arts and Sciences, and a member of the American Philosophical Society.

It is not difficult to read between the lines of the foregoing

sketch that Prof. Pettee was one of those who devote versatile accomplishments and varied experience to the service of their fellows, rather than their own benefit. He might have won a more conspicuous fame, if ambition had been his chief motive-power. But he was content, as a teacher, to impress and express himself through the inspiration of others, and, as a critic and editor, to perfect the utterances of others, without demanding credit for himself.

An instance, not mentioned in any of the published accounts of his work which have come to my notice, may serve as an illustration. Some years ago, I unhesitatingly named Prof. Pettee to the publishers of the "Standard Dictionary" as the most competent person to prepare for that publication the definitions of mining and metallurgical terms. He accepted the task; and his consequent work, though buried beyond special recognition in a mass of anonymous contributions, is a monument of accurate learning and critical acumen, which will benefit the professional student for many years to come.

But all this is only a prelude to my acknowledgment of the benefit unselfishly conferred by Prof. Pettee upon the American Institute of Mining Engineers. He became a member of the Institute in 1871, the year of its foundation; and, in all the years that followed, his only formal contribution to its *Transactions* was the Biographical Notice of Prof. Byron W. Cheever, his colleague in the faculty of the University of Michigan.[1] Yet no one has contributed to the Institute more valuable, effective, generous aid than he. I do not refer only to his service as Vice-President in 1888 and 1890, but to labors far more extended, continuous and important, though less openly recognized.

At a very early day,—before 1884, when I became Secretary of the Institute,—Prof. Pettee was known to my friend and predecessor, Dr. Drown, as one who could be relied upon for ready and skilful assistance. Later, I came to regard him as, by reason of accomplishments, exact knowledge and sound critical judgment, the best "all-round" expert among the members of the Institute; and, many years ago, I engaged him to aid me in my editorial work by making the final revision of the sheets of the Institute volumes of *Transactions*.

[1] *Trans.*, xvi., 888.

I may be pardoned for stating somewhat at length the nature of this aid. It has been the aim and pride of the editor of these volumes to make them, as nearly as possible, perfect in style and accuracy. The manuscripts and drawings have been prepared with care for the printer or engraver; the proofs of both have been revised by proof-reader, editor and author; the preliminary pamphlet-edition has been similarly submitted to repeated revision; and, after all this, the sheets for the volume have been ordered to press. But the last touch of perfection, it seemed to me, could not be reached without a supplementary critical examination by a wholly fresh mind, not biassed by any previous acquaintance with the text. Consequently, I engaged Prof. Pettee to make "the ninth revision." After everybody else had done his work, the "signature" of 16 pages was put on the press, and copies were printed and sent to Prof. Pettee; after which the forms were taken off the press again, to await his verdict, before the final printing of the edition. Concerning the value of this verdict, I cannot do better than quote the acknowledgment contained in my preface to vol. xxix. (July, 1900):

"The steady growth of the Institute in membership, and the increased number of contributions from members in distant lands, add every year to the labor and time required for the production of a volume approximately free from errors of the pen or press. . . .

"In this connection, I would here acknowledge the efficient assistance of Prof. W. H. Pettee, of the University of Michigan, who has, for many years past, given to the sheets of the volumes of *Transactions* their final revision. The importance of such a supplemental scrutiny is shown by the fact that, of several thousand papers thus submitted to Prof. Pettee, probably less than a score have escaped without some minute correction."

This preface did not go to Prof. Pettee for critical examination; and when he first saw it in the published volume, he wrote me a characteristic letter, in which, after a modest disclaimer of my praise of his work, he pointed out that my use of the term "several thousand" was not quite warranted by a careful count!

But the mere general term "revision" does not convey an adequate notion of the scope and thoroughness, in his hands, of such a labor; and I take this opportunity, not only in justice to him, but also because I think it will be interesting and instructive to others, to set forth more fully the nature and extent his invaluable service.

Our printer is expected to furnish a "clean" proof, free from broken letters or other typographical errors, and in accordance with the editorially-prepared copy furnished to him. The editor or the author may make changes in this proof, in correction, either of errors overlooked by the printer, or of errors in the copy, overlooked by them, or for the improvement in clearness, style or arrangement of the text. Changes of this last class frequently suggest themselves only after the text has been put in print, and can be read as one reads a book. It is an interesting psychological fact, however, that, as a rule, authors themselves do not spontaneously suggest such changes. To them, the text represents what they had in mind when they wrote it; and they do not readily realize how it may strike an intelligent reader who has no other means of knowing their full meaning.

But after all such revisions, the final proof may contain: (1) some errors of any kind, strangely overlooked by previous revisers; (2) errors resulting directly from previous corrections, the collateral consequences of which were not seen; (3) errors in references and quotations, tables and formulas, or the spelling of proper names; (4) errors in scientific theory or inconsistencies of statement, which have escaped previous notice; and (5) the most provoking errors of all, constituting the terror of authors and editors, namely, those which result from accidents to the "forms" of type, after they have been finally ordered to press,—as, for instance, when a few letters fall out of an imperfectly-locked "form," while it is being carried to the press, and the foreman remedies the disaster in an extempore, amateur fashion, rather than confess negligence and impede business by referring the whole thing back to the editor.[2]

Now, all the revisions indicated in the foregoing statement were repeated in Prof. Pettee's "ninth revision." And he was

[2] Years ago, Mr. Richard Grant White made piteous complaint (I think in the *Atlantic Monthly*) of this cause of error, which had rendered him responsible for a blunder severely criticized by the reviewer of his book. Such disasters do not happen in these days of "linotype"-work, in which each line of type is a single mass; but linotype-work, though cheaper, is excluded in the publication of our *Transactions*, because it does not permit the free correction of the text. We have everything set up in type, in the old-fashioned way; and we keep the type standing (often more than a year) until, after all possible corrections, the sheets of the volume have been printed.

competent for every one of them. He would detect a broken or inverted letter, a column of figures that did not "add" right, a mistake in a chemical formula or algebraic equation, an incorrect reference or quotation, a blunder in a foreign tongue, or a logical absurdity, obscurity or contradiction—all with equal certainty and precision. Backed by the great library of the University of Michigan, he was absolutely indefatigable in following the trail of the smallest question involving reference, quotation, or statement by one author of the views of another. It was, indeed, a startling revelation to me that, of the passages marked by authors as quotations, he found so many which gave the quoter's notion of the meaning, instead of the exact words of the original.[3] In this, as in all other particulars of the ethics of authorship and scholarship, Prof. Pettee was an unerring and uncompromising authority.

A few instances, taken at random from his letters to me, will illustrate the foregoing statements. It was his custom to report to me all except the most obvious and trivial corrections which he had made, giving his reasons therefor, and, in any case of reasonable doubt, referring back to me the signature concerned, and notifying the printer not to proceed until after receiving my decision. With this explanation, the following extracts may be appreciated:[4]

On p. —, at line —, I find the name Thompson. On p. —, line —, apparently the same party is mentioned as Thomson. Have no means of knowing which is correct. Have notified printer to await your decision.

On p. —, line —, the accented "e" is wrong. In that word the "e" carries no accent.

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[3] I believe it was the *Saturday Review* which, in criticizing the history of Mr. Froude, who had "quoted," not the full text, but his own summary, of certain letters bearing upon the case of Mary, Queen of Scots, uttered the famous significant and scathing criticism, "Mr. Froude does not understand the meaning of quotation-marks." I fear that many writers of professional papers are open to a similar charge—though they may not be equally blameworthy, since they may not be sinning against equal light.

[4] These passages are not always quoted verbatim, being often summarized or re-stated with omission of the context. But every one of them correctly represents the substance of an actual communication.

On p. —, lines — and —, the name of this foreign firm is given as ——; on p. —, as ——. On consultation of U. S. consular reports, I find the name of the firm, as there given, to be ———; and assuming this to be correct, I have corrected both p. — and p. — accordingly.

The statement on p. —, line —, is inconsistent with the previous statement, p. — line —. To save you time, I have written to the author, and will hold this signature back from publication until I have heard from him.

What is the scale of Fig. — on p. —? Judging from statements on p. —, I infer that it is a quarter-inch to the foot. I send the signature to you, and have notified the printer to await your decision.

On p. —, the author's statement is obscure, and I have been unable to decide exactly what he means. Judging from p. —, and from the conditions of the problem as stated, I conjecture. . . . But I leave you to decide whether you will telegraph him for an explanation, or let his statement stand, to puzzle other readers, as it puzzles me, or alter it on your own responsibility, or manage to omit it altogether (with small loss to anybody). Meanwhile, I have notified the printer not to put this signature to press until he receives orders from you.

The quotation on p. — has given me a good deal of trouble. Our University library possesses copies of two editions of the work cited; but in neither of them can I find this passage, though the view which it states might fairly be deduced from the statements of both. Perhaps it actually occurs in a later edition, not in our library. Meanwhile, however, in pursuing this matter, I have found in another work, —— 's ——, the exact words of this quotation, given, however, as the author's summary of the opinion stated in the earlier work. It looks as if the writer of this Institute paper had taken this summary, put it in quotation-marks, and credited it to the earlier work, which, in fact, he had never consulted. If this be the case, there is, of course, no doubt as to what literary honesty requires. I refer the matter back to you, and have notified the printer to await your decision.

Table —, p. —, has given me a good deal of trouble. The total of column — is not correct; and the "averages" seem to have been calculated upon other data, not given in the table. I refer this difficult problem back to you.

To-day, I returned signature — to the printer, with my approval; and I notify you of the fact, in order to have the pleasure of saying to you that in these sixteen pages I have had to make no corrections.

The foregoing typical samples will doubtless suffice for my

purpose, though I could increase indefinitely, by additional citations, their scope as well as their number. When it is remembered that this sort of patient, careful, and varied work was carried on by Prof. Pettee, year after year, with unfailing zeal and keenness, without thought of public recognition, and without any other motive than the conscientious desire and the deep delight of doing useful work thoroughly, it may be possible to measure, to some extent, his merit and our debt of gratitude to him.

But there is a further circumstance, which, in my judgment, should emphasize our affectionate memory of him. For a number of years before his death, Prof. Pettee was subject to periodical acute attacks of some chest-trouble, which temporarily incapacitated him for work. Members of the Institute who have enjoyed his genial presence on Institute excursions will remember how he was obliged to sit down, while the rest went on, and endure a spasm of pain and weakness, before he could rejoin the party. Such a disease was of course a warning, which might well have warranted him in withdrawing himself from all such responsible duties as he could with fairness and honor decline. Yet he continued, in spite of this threatening disability, to render to the Institute the service for which he was almost uniquely qualified; and, in spite of all interruptions and disabilities, he kept his part of the work "up to date," without noticeable lapse. His trouble, however, gradually increased; and in August, 1903, he wrote me that an expert examination had diagnosed his disease (previously called a muscular rheumatism) as *angina pectoris*, and that, although he might live, and continue to do light work, for years to come, he thought it but right to notify me of the precarious tenure of his strength, and his possible sudden failure in the midst of his labors for me and for the Institute. I need hardly add that my reply assured him of my desire that he should go on with those labors as long as he could, and my opinion that this was just the kind of "quiet occupation" which his physician had recommended as better for him than a complete retirement from all occupation. A little later, I wrote him, during an illness of my own,

"We must both watch our machines, and get out of them all the work they can still do, without complaint and without despondency. I am inclined to think

that this mood will result, in your case as in mine, in the maximum of effective life-work—all the more effective, because, in our years of enforced moderation, we put more mature judgment, and less reckless activity, into what we are still permitted to do."

Thus I preached; for preaching is easy. It was he that practised—needing not my sermon.

He died May 26, 1904, discharging, up to the night before, his regular duties in the University. Almost the last hours of his life were spent over the sheets of the Institute *Transactions;* and after he had passed away, I received from his family the results of his latest work, so perfect in scholarly accuracy and critical judgment as to give no hint that they had been wrought with failing strength and under the shadow of death. The delicate machinery of his mind was like that of a watch, which, as it ran down, kept time to the last tick.

The watch needs only to be started again; and I, at least, cannot conceive that the unimpaired reason, manifold knowledge, balanced judgment, dauntless perseverance and loyal affection of such a man as William H. Pettee can be less capable of renewed activity—that a life which had moved so long in so straight a line, had no momentum. Apart from all subtilties of philosophic or theologic doubt, it seems to me that my friend might well have uttered as his own the famous retort: "I do not undertake to say that *you* are immortal; but I know that *I* am!"

Prof. Pettee was married, July 8, 1874, to Sybil Clarke, of Newton Upper Falls, Mass., who, together with a daughter, Mrs. Earle W. Dow, survives him.

The Genesis of the Diamond.

BY GARDNER F. WILLIAMS, KIMBERLEY, SOUTH AFRICA.

(Lake Superior Meeting, September, 1904.)

CHEMICALLY, the diamond is composed of the element carbon in its pure crystallized state. The diamond crystallizes in the isometric system, and the most common forms are the octahedron and dodecahedron, while the (24-sided) tetrahexahedron is not uncommon. Cube diamonds with beveled edges representing the combination $\infty\, 0\, \infty$ and $\infty\, 0\, 2$ are occasionally found in the Bultfontein and Premier (Wesselton) mines at Kimberley, South Africa. The diamonds from various mines have distinctive forms of crystallization, or variations of the same forms, so characteristic that those familiar with South African diamond-mines and their products can determine positively from which mine any given parcel of diamonds has been obtained. It is not always possible to determine the source of each individual diamond, for similar stones are occasionally found in different mines; but these are exceptions to the rule. There is a difference in the luster, shape or crystalline form of the diamonds from the various mines that gives each mine some distinctive characteristic. In one mine nearly all of the crystals are sharp-edged octahedrons, while in another, dodecahedrons with rounded faces predominate. One might give no end of peculiarities of the diamonds from the various mines; but it will suffice for the purposes of this paper to state the fact that such distinctive characteristics do occur.

From this observation it may be concluded that the diamonds in the mines of the Kimberley district, which occupies a small area (see Fig. 1), did not have a common origin.

The diamond is the most impenetrable of all known substances, and will scratch any other stone, or the hardest steel. It is a very strong reflector of light, and refracts incident rays more than any other substance except crocoite.[1]

[1] *Table of Indices of Refraction*, in Dufrenoy's *Traité de Mineralogie.*

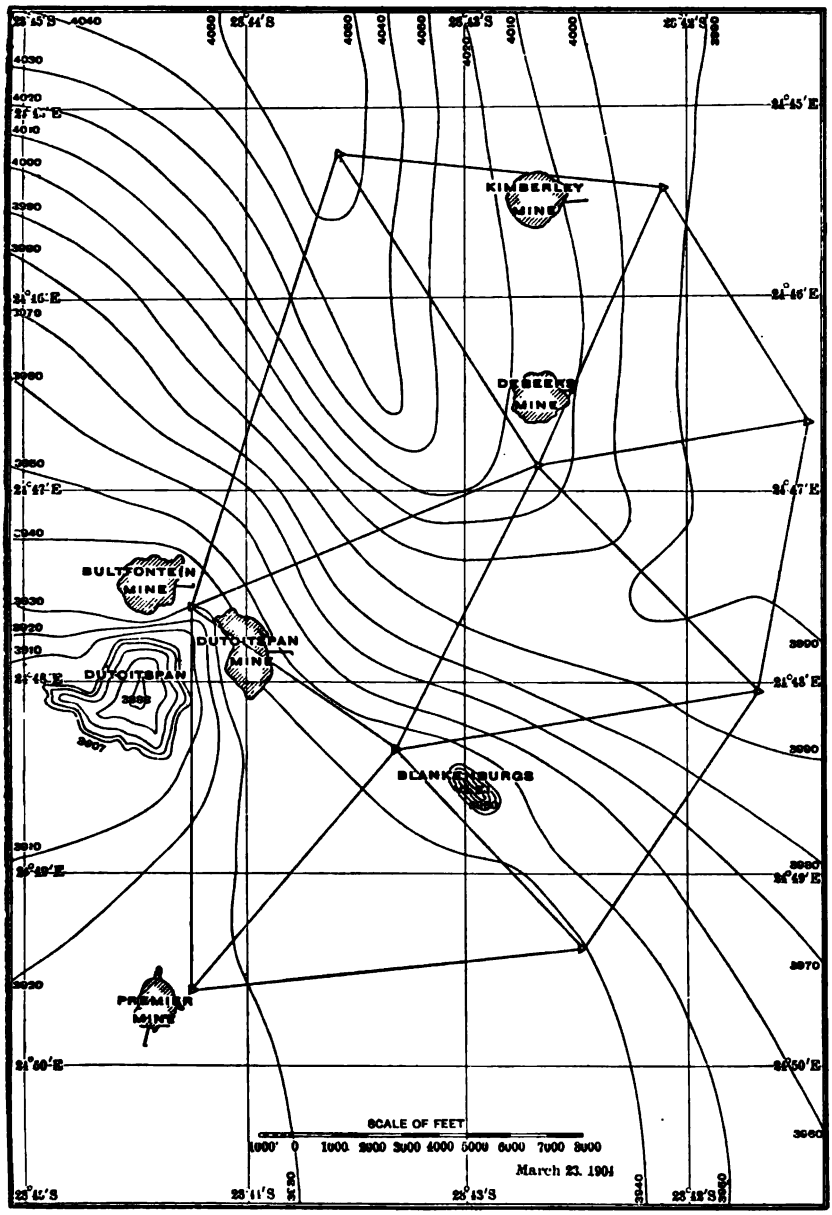

FIG. 1.—DIAMOND MINES IN THE KIMBERLEY DISTRICT, SOUTH AFRICA.

While crocoite is the only mineral that exceeds the dispersive power of diamond to dissolve white light into rainbow-tints; in its powers of reflection, refraction and dispersion taken together, the diamond is unmatched.[2] It is highly phosphorescent, and even the blackest diamond is transparent to the X-ray. It is insoluble in all acids, and can easily be burned and converted into carbon dioxide. However, it is noteworthy that the diamond is a non-conductor of electricity, while graphite and amorphous carbon, substances so closely allied to it in chemical composition, are good electrical conductors. By the application of friction, the diamond can be positively electrified, but it very soon loses its electricity. The diamond is easily cleaved in planes parallel to the octahedral faces. Pieces may be easily broken from the facets of a cut stone by striking it with a hard substance.

So much, it may be claimed, is known about the physical properties of the diamond, but how the diamond has been formed or crystallized is a question still debated by scientists.

Upon the information at hand, it may be assumed that all diamonds found prior to the discovery of the Kimberley pipes, or craters, came from alluvial deposits, and had been washed down by the disintegration of the original matrix. Such was the character of the formation in which the noted diamonds of India were found, which is described as a layer of broken sandstone, quartz, jasper, flint, and granite, interspersed with masses of calcareous conglomerate, the whole being about 20 ft. thick and covered with a few feet of black "cotton soil." Here were the great mines of Gani-Coulour and Gani-Parteal, whence came the Koh-i-nûr (793 carats), the Great Mogul (787.5 carats), the Regent (410 carats), and many other historical stones.[3]

The Indian mines were noted more for the size and purity than for the quantity of the gems they produced. There had been no considerable production of diamonds outside of the Deccan fields until the discovery of diamonds in Brazil in the year 1728. Here in the province of Minas Geraes rich beds were opened in an alluvial deposit of clay, quartz pebbles and

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[2] Feuchtwanger's *Treatise on Gems.*

[3] *Voyages en Turquie, en Perse et aux Indes,* Tavernier, Paris, 1676.

sand charged with iron oxide.[4] In many places the diamond-bearing strata were buried under 30 ft. or more of alluvial detritus. These deposits occurred along the rivers, up the ravines to the ridges and plateaus, where conglomerate-beds were reached from which the deposits in the rivers had been washed. The conglomerate was chiefly itacolumite, a micaceous sandstone. The sandstone being a fragmental sedimentary rock was not the original matrix of the diamond. Probably, when the sandstone was being formed, the diamonds were washed down with the detritus, and became imbedded in it.

Diamonds have also been found in Borneo; New South Wales; British Guiana; in the gold-deposits of the Ural mountains, Australia and California; along the Vaal river in South Africa; and in many other localities, but always in alluvial deposits.

Before the discovery of the mines at Jagersfontein and Kimberley, which occurred between August, 1870, and July, 1871, there is no record that diamonds had been discovered in volcanic pipes or craters; their occurrence having always been in alluvial or sedimentary deposits.

A few years ago, diamonds were found in the battery-mortar of a mill at Klerksdorp in the Transvaal, which was crushing gold-ore from a conglomerate-reef, similar to the Witwatersrand reefs. It had occurred to me that either the wash from a diamond-bearing pipe had been mixed with the detritus when the conglomerate-beds were formed; or, that a diamond-bearing pipe penetrated the conglomerate-strata and its contents were mined and sent to the mill along with the gold-ore. This remarkable occurrence of diamonds has been explained to me by an old Kimberley miner who was on the ground at the time the diamonds were found.

In his opinion some of the top soil or wash was mined with the gold-bearing conglomerate and the diamonds came from this alluvial deposit. The finding of these diamonds in the battery-mortar, as above described, is well authenticated. The diamonds were of a greenish color.

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[4] The Diamond Fields of Brazil, *Report of U. S. Minister Bryan*, 1899, conveying report of Secretary of Legation, Dawson. *A Treatise on Gems*, Lewis Feuchtwanger, M.D., 1867. *An Account of Diamonds Found in Brazil*, James Castro de Sarmento. *Travels in South America*, J. J. von Tschudi.

Coming now to the occurrence of diamonds at Kimberley, I may preface my remarks by saying that my experience with the mines at Kimberley dates back nearly twenty years, seventeen of which have been passed in the management of them.

The diamonds occur in a rock commonly known as "blue ground," filling the craters of extinct volcanoes. This rock was described by Professor Henry Carvill Lewis as a porphyritic volcanic peridotite of basaltic structure,[5] which he named "Kimberlite." It must be designated as a breccia. There is no doubt that the blue ground is of volcanic origin, and was forced up from below; it consists of olivine with fragments of other rocks. I am of the opinion that the craters were filled by aqueous rather than igneous agencies, possibly by something in the nature of mud-volcanoes.

It is a noteworthy fact that all the craters were filled just even with the surface of the surrounding country. Would this have been the case if the pipes were of igneous origin? I think not.

It may be claimed that the surface of the country, as it existed when the craters were filled with the diamond-bearing breccia, was not the same as at present, but that it has been denuded or washed away, or has been removed by glacial action. There is, however, not the least particle of evidence to bear out such a contention. If the country-rock and diamond-bearing ground had become decomposed and been washed away, then diamonds would have been found in the "wash," or in ravines and water-courses in the vicinity of the mines. Such is not the case; and no diamonds have been found in alluvial soil nearer than the Vaal river, about 20 miles distant; and these diamonds are totally different in character from the "mine" stones.

The Vaal River diamonds did not have their origin in the Kimberley mines. The occurrence of well-rounded and, at times, polished boulders and small pieces of rock, is proof that other than igneous action was necessary to produce them.

Professor Bonney says that "the idea that they have been rounded by a sort of cup-and-ball game played by a volcano may be dismissed as practically impossible."[6] He refers to the

[5] *The Matrix of the Diamond*, Henry Carvill Lewis, M.A., F.R.S., at a meeting of the British Association at Manchester, August and September, 1887.

[6] *Proceedings of the Royal Society*, vol. lxv., 1899.

Dwyka conglomerate-bed as a possible source of these boulders. A conglomerate-bed exists in the Kimberley strata between the shale and the melaphyre (or olivine-diabase of Stelzner), and is between 300 and 400 ft. below the surface. This conglomerate is from 3 to 10 ft. thick, as determined in the various shafts in the Kimberley mines. The rounded stones in the mines did not come from this bed, and are wholly unlike the stones in the conglomerate. Personally, I do not favor the cup-and-ball theory, and would not give it a second thought were it not for the fact that the diamond-bearing ground as it is found in the mines shows such a mixture with the country-rock that some process of nature must have stirred up and thoroughly mixed the contents of these great craters. I cannot comprehend how this result could have been brought about in an igneous volcano. There would have been overflows of the diamond-bearing rock which would have been found in the vicinity of the mines. No such deposits have been found, and I do not believe that they exist.

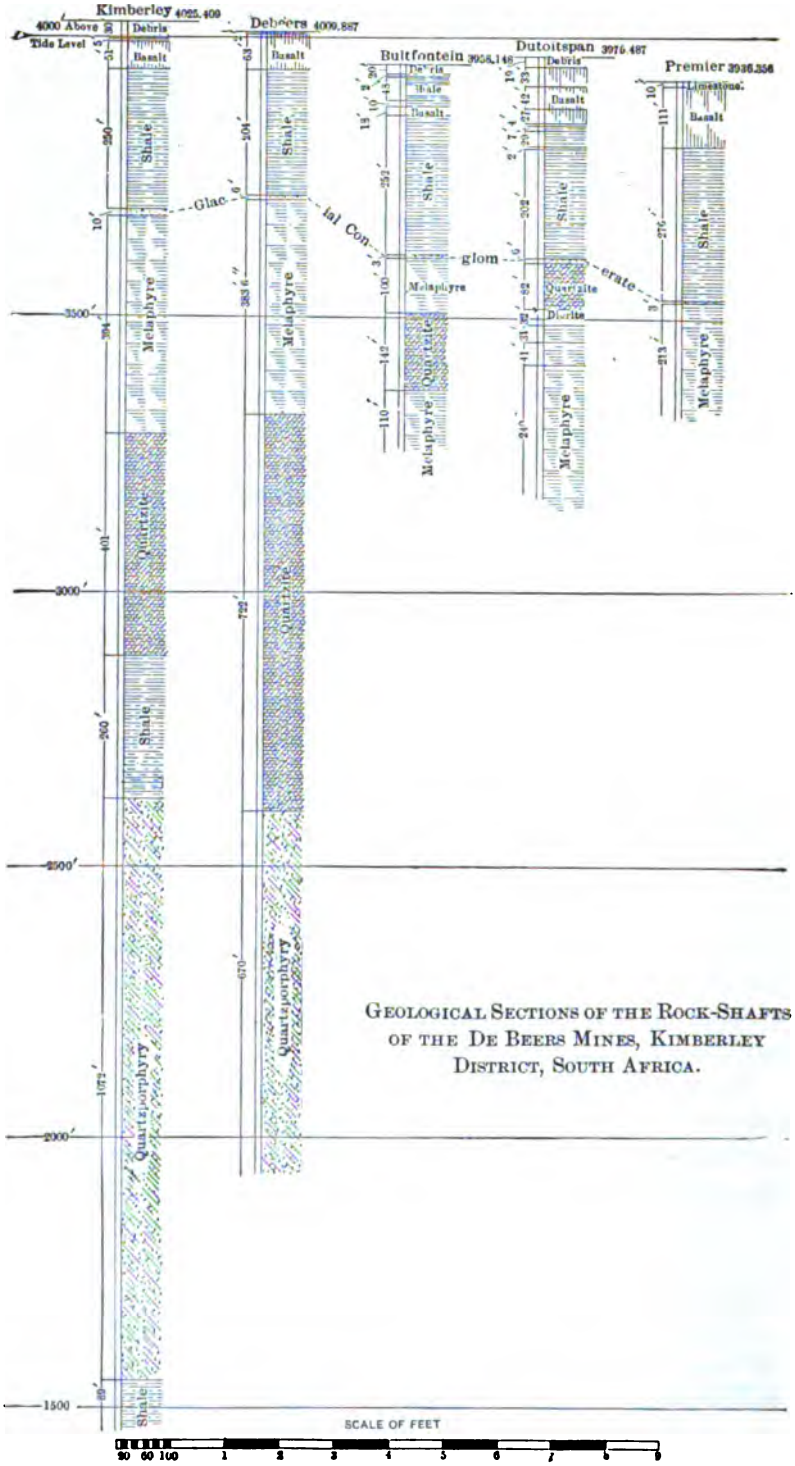
It is much easier to reconcile existing conditions to the aqueous or mud-volcano theory (especially if the mud was accompanied by large quantities of gases which, on nearing the surface, escaped while the mud receded) than to an igneous theory.

There must have been innumerable eruptions and explosions to account for the inclusion of the surface shales and fragments of the country-rock in the diamond-bearing peridotite. The frequent occurrence of these eruptions would, in a measure, solve the problem as to the manner in which the fragments of rocks varying in size from pebbles to boulders, some with polished surfaces, became, as it were, waterworn.

The evidence of the movement of the diamond-bearing rock after solidifying is indicated by the slickensides and striated surfaces of the country-rocks at their junctions or contacts with the kimberlite. Large sheets of calcite are frequently found at the junction of these rocks, which have taken the form of the striæ. Beautiful calcite crystals and transparent pieces of doubly-refracting, or Iceland spar, are of frequent occurrence.

There is conclusive proof that the diamonds in the South African mines are not found in their original place of crystallization, as is shown by the frequent occurrence of broken crystals imbedded in the hard kimberlite. The geological

FIG. 2.



GEOLOGICAL SECTIONS OF THE ROCK-SHAFTS OF THE DE BEERS MINES, KIMBERLEY DISTRICT, SOUTH AFRICA.

March 23, 1904

strata of the rocks which surround the diamond-bearing pipes of the Kimberley district are shown in Fig. 2.

Concerning the discussion of the genesis of the diamond, Sir Isaac Newton's opinion was that it was of vegetable origin and combustible; but it was not until 1694 that the combustibility of the diamond was actually proved by the famous burning-glass experiment of the academicians of Cimento.

Lavoisier, Guyton de Morveau and others determined that the diamond was converted into carbonic dioxide by burning. The experiments of Sir Humphry Davy, in 1816, showed that the diamond was almost pure carbon. These experiments have been confirmed by Dumas, Stas, Friedel, Roscoe, and other eminent chemists, who have fixed with extreme precision the composition of the diamond to be pure carbon in crystalline form. The late Dr. W. Guybon Atherstone was one of the first scientists to deal with the occurrence and genesis of the diamond in the Kimberley mines. Being a resident of the Cape Colony, he made frequent visits to the diamond-fields and made personal investigations.[7]

"For a substance to crystallize," he says, "its molecules must be free to move. . . . The diamond, we know, is neither soluble nor fusible. It is the element carbon crystallized, and is consumed by heat. How, then, could it survive as a crystal in the center of a volcano? The key to solve this mystery was placed in my hands over half a century ago by one of the greatest philosophers of the age, whose lectures I had the privilege of attending. . . . 'Hold out your hand,' said Faraday, at the close of the lecture that fairly electrified the world of science, as with a loud hiss a snowy substance, burning like a coal, but in reality intensely cold, escaped into the palm of my hand from the strong iron vessel, in which, with a pressure of fifty atmospheres, he had liquified carbonic acid gas, —the very gas resulting from the combustion of the diamond." . . . "In the carbonic acid gas generated from the carbonaceous shales by heat, and interspersed as gas bubbles in the cavities of the viscid, ferruginous amygdaloid and in the admixture of steam, lava and ashes known as the "Kimberley blue," reduced to the liquid state by the enormous pressure in the subaqueous volcano, we have the constituents of the diamond in a form admitting of crystallization, and the subsequent absorption of its oxygen by the iron always present in its containing walls during long intermittent periods of volcanic inactivity."

In this presentation Dr. Atherstone dogmatically puts the carbonic acid gas evolved from the carbonaceous shales into the cavities of the amygdaloid (presumably the melaphyre, which is the only one of the encasing rocks of the volcanic pipes that is amygdaloidal). This gas is then reduced by pres-

[7] *Geological Magazine*, vol. vi., p. 208, May, 1889.

sure to a liquid state, in which form, as he thought, the carbon admitted of crystallization. He then absorbed the oxygen of the carbonic acid by the iron in the containing-walls of the craters. As the melaphyre existed before the volcanoes burst through it, it is more than probable that the cavities, which existed in it at the time it was erupted, were filled with agate and calcite, which they now contain, before the diamond-bearing ground was forced up through it. If the theory above given had any foundation in fact, one of two results must have happened, viz.:—either the resultant diamonds would have been enclosed in the amygdaloidal rock, or the diamonds must be formed in the "blue" in their perfect state. Both of these assumptions are contrary to facts. As to the derivation of the necessary carbon from the carbonaceous shales surrounding the mines, it will be made clear subsequently that this assumption is not justified.

Professor Lewis alleged that the diamond is the result of the intrusion of igneous rocks into and through the carbonaceous shales.

He says:[8]

"Perhaps the most interesting chemical observation concerning the blue ground was that made by Sir H. E. Roscoe. He found that on treating it with hot water an aromatic hydrocarbon could be extracted. By digesting the blue ground with ether, and allowing the solution to evaporate, this hydrocarbon was separated and found to be crystalline, strongly aromatic, volatile, burning with a smoky flame and melting at 50° C.

"That the rock was a true lava and not a mud or ash is indicated by the fact that the minerals and their associations are those characteristic of eruptive ultra-basic rocks."

Professor Lewis further says:

"The kimberlite is shared by no other terrestrial rock. In structure it resembles meteorites of similar composition. If the ground-mass of kimberlite were replaced by native iron, it would be nearly allied in both structure and composition with meteorites known as chondrites."

The "Ava" meteorite, which fell in Hungary in 1846, contained graphite in cubic crystalline form which Gustav Rose thought was produced by the transformation of diamonds. Later Weinschenk found transparent crystals (diamonds) in the Ava meteorite. Minute diamond crystals and graphite have been found in the meteorites from Cañon Diablo, Arizona.

[8] *The Matrix of the Diamond.* Professor Henry Carvill Lewis, p. 52.

Professor Lewis advanced the theory that probably the diamonds came from the hydrocarbon which was contained in the fragments of carbonaceous shales distributed through the blue ground, but the inclusion of carbonaceous shales in the blue ground can hardly be reconciled with Professor Lewis's conclusion "that the rock was a true lava."

If the diamond is the result of the intrusion of igneous rocks into and through the carbonaceous shales, why do not all pipes composed of kimberlite contain diamonds? And why do diamonds exist in some mines, such as those in the Pretoria district, where no carbonaceous shales are to be found?

Professor Molengraaff, formerly State Mineralogist to the South African Republic, discusses[9] the genesis of the diamond, and says that the theory of the formation of diamonds during the ascension of the blue ground from carbon borrowed from the carbonaceous shales, was, in his opinion, weak.

"In the Pretorian beds, as well as in the formations underlying these, strata containing any notable quantities of carbon were nowhere to be found in the Transvaal; so that the conclusion might safely be drawn that the igneous blue ground, in forcing its way from great depths towards the place where it was found, could not borrow any carbon from the surrounding strata in order to convert it into diamonds."

In Bohemia a rock occurs which contains every mineral known in the blue ground of Kimberley, except diamonds. On my visit to the Mining Academy at Freiberg, Saxony, a few years ago, Dr. Stelzner, Professor of Geology, showed me two cases containing these minerals, and in every instance the Bohemian minerals corresponded with those from Kimberley, except that the case of Kimberley minerals contained a few small diamonds which had been presented to the Academy.

Both the aqueous and igneous theories of the origin of the Kimberlite have had able supporters, among those of the former being Stanislas Meunier,[10] M. Chaper,[11] and later Professor Gar-

[9] *A Monograph on the Diamonds at Rietfontein, near Pretoria, in the Transvaal.*

[10] *Composition et origine du sable diamantifère du Du Toits Pan, Afrique Australe—Comptes rendus de l'Academie des Sciences de Paris*, vol. lxxxiv., No. 6, p. 250. *Examen mineralogique des roches qui accompagnent le diamant dans les mines du Cap de Bonne Esperance—Bulletins de l'Academie Royale de Belgique*, 3d series, vol. iii., No. 4

[11] *Note sur la région diamantifère de l'Afrique Australe*, Paris, 1880.

nier and Sir William Crookes.[12] The igneous theory is strongly supported by Professors Lewis,[13] Molengraaff,[14] and Stelzner.[15] My own opinion is that the aqueous theory is the less assailable.

Concerning the origin of the blue ground, assuming that it is not the original matrix of the diamond, I find the following weak points in the igneous theory.

1. As already observed, it is impossible to account by the igneous theory for the water-worn boulders found in the blue ground.

2. The experiments of Herr W. Luzi,[16] of Leipsic, in the production of artificial figures of corrosion upon the surfaces of rough diamonds, are most interesting in the light which they throw on the crystallization and the probable matrix and genesis of the diamond. Until lately, the only appearance of corrosion upon the surface of rough diamonds was the regular, triangular negative pyramids which were produced through heating the diamond in the open air or under the oxygen flame.

Herr Luzi discovered that the breccia (kimberlite) from the South African mines, when in a molten condition, possesses the property of absorbing the diamond or of changing its shape.

The following is a translation of the description of his experiment:

"A small quantity of blue ground was melted in a crucible placed in a Fourquinon-Leclerq furnace at a temperature of 1770°, which was the highest temperature attainable. A diamond with perfectly smooth natural faces was submerged in this molten mass. A further quantity of blue ground was added to the contents of the crucible until it was completely filled. A tightly fitting cover was placed on the crucible, which was again exposed for 30 minutes to the greatest heat attainable. When the crucible was cooled, the diamond was removed, and found to be covered with irregular oval and half-round grooves of various depths. In one experiment, the diamond was found to be deeply eaten away on one side."

Some of these partly absorbed diamonds, upon which Herr Luzi experimented, are deposited in the mineralogical museum of the Leipsic University.

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[12] A lecture before the Royal Institute of Great Britain, June 11, 1897.

[13] *The Matrix of the Diamond*—Henry Cavill Lewis at a Meeting of the British Association at Manchester, August, 1887.

[14] *The Occurrence of Diamonds at Rietfontein*, G. A. P. Molengraaff.

[15] A lecture by A. W. Stelzner before the Isis Society, in Dresden, Saxony, April 20, 1893.

[16] Ueber künstliche Corrosionsfiguren am Diamanten, *Berichte der Deutschen Chemischen Gesellschaft*, xxv., p. 2470 (1892).

Owing to the cost of the material to be experimented upon, however, Herr Luzi was unable to determine positively what chemical action took place during the time the diamonds were heated in the complicated silica-flux. The fact that diamonds can be absorbed by being placed in molten blue ground tends to prove that the blue ground was not thrust up through the earth's crust in a molten state.

If the diamond is unable to withstand the corroding influence of the silica-magma at the comparatively low temperature given above, how could it possibly have retained its forms of crystallization and perfect faces at the far higher temperature and pressure which must have existed under the igneous theory?

It seems a pity that Herr Luzi did not state the exact weight of the diamonds upon which he experimented, both before and after his experiments. The burning or absorption of the diamonds in its matrix is a strong argument against the contention that the blue ground was once a molten lava. If a diamond placed in a graphite crucible containing melted blue ground, which is subjected to a temperature of only 1770° R., changes its shape, could diamonds be found perfect in shape, without a flaw, and with clear transparent faces, so smooth that they have the appearance of having been polished?

3. Some years ago a diamond weighing 28.5 carats, found at Kimberley, attracted the attention of the valuator. Its external surface was smooth and crystallized, showing no other mineral except the diamond itself, but the interior was white and not transparent. Noticing this peculiar appearance, the valuator broke the stone in order to satisfy his curiosity, and found that a small perfect octahedral diamond was enclosed in the center of the larger stone. Nor was this all. There were flakes of a white mineral, not diamond, attached to the fragments of the broken diamond. In appearance the flakes were white, translucent and crystalline, and about as hard as steel. When heated in a closed tube, moisture was given off. It fused readily on platinum wire to a white bead. A few grains of this white mineral were collected, and by analysis it proved to be apophyllite, a silicate of lime and potash with 16 per cent. of water.

If a mineral which is fusible at the ordinary temperature

obtained with a blow-pipe, and which contains 16 per cent. of water, was formed at the same time that the diamond crystallized, it is certain that this did not take place under an enormously high temperature. How, then, one may ask, did the apophyllite become a part of this diamond?

Herr von Tschudi[17] describes a beautiful crystallized Brazilian diamond in the center of which was a leaf of gold. He obtained the information from Dr. Mills Franco, who claimed that there was no doubt or deception as to the identification of the gold.

Occurrences of this nature tend to veil in additional mystery the genesis of the diamond.

4. Professor T. G. Bonney[18] obtained from the Newlands mines, 40 miles northwest of Kimberley, specimens of a coarsely crystalline rock studded with garnets, technically called holocrystalline, and allied to eclogites. At a meeting of the Royal Society,[19] he presented his conclusions:—

"The blue ground is not the birthplace, either of the diamond or of the garnets, pyroxenes, olivine, and other minerals, more or less fragmental, which it incorporates. The diamond is a constituent of the eclogite, just as much as a zircon may be a constituent of a granite or a syenite." . . . "I had always expected a peridotite (as supposed by Professor Lewis), if not a material yet more basic, would prove to be the birthplace of the diamond.

"Can it possibly be a derivative mineral, even in the eclogite? Had it crystallized out of a more basic magma, which, however, was still molten when one acid more was injected and the mixture became such as to form eclogite? But I content myself with indicating a difficulty and suggesting a possibility; the fact itself is indisputable: that the diamond occurs, though rather sporadically, as a constituent of an eclogite, which rock, according to the ordinary rules of inference, would be regarded as its birthplace."

Professor Bonney's statement that diamonds occur in the eclogite of the Newlands mine caused me to examine the eclogite which is found in all the mines at Kimberley, and has always been treated as waste rock and thrown away. There are tons of it lying about the Kimberley mines. I have examined hundreds of pieces of this rock, but never found a diamond; nor have I ever heard of a diamond being found in it by any one during the many years that these fields have been

[17] *Travels in South America*, by J. J. von Tschudi.

[18] *The Parent Rock of the Diamond in South Africa*, Professor T. G. Bonney.

[19] *Proceedings of the Royal Society*, vol. lxv., July 27, 1899.

worked. I caused about 20 tons to be collected and sent to a test-plant, where it was crushed and afterwards jigged; but it contained no diamonds. Surely, if one could find diamonds in the eclogites of a poor mine like the Newlands, the total diamond-yield of which was only a few hundred carats, one would naturally expect to find them in the eclogites from mines in the vicinity of Kimberley, which are so rich.

5. Sir William Crookes and others have mentioned diamonds which burst or explode on being brought to the surface; and Sir William says it has been "conclusively proved that the diamond's genesis must have taken place at great depths under enormous pressure. The explosion of large diamonds on coming to the surface shows extreme tension."

Professor Lewis says that Kimberley diamonds have been found sometimes to have optical anomalies due to strain. Fizzan thought this strain to have been caused by the unequal distribution of heat during cooling; but Jannettaz[20] holds that the strain is due to compressed gas in the interior of the crystal.

I have found that the light-brown smoky diamonds are the only ones which crack on being brought to the surface; but even these remain intact if kept in a moist place. In the days of open-cut working, when a smoky or light-brown diamond was found, the digger placed it in his mouth, where he kept it until he offered it for sale. The buyer placed it in a raw potato, in which it was shipped to Europe. The temperature of the ground in which the stone was found would, as a rule, not exceed 70° F. The temperature of the diamond would be raised to 98° F. while in the digger's mouth. If, however, the stone was kept in a dry place, even at a lower temperature, it would crack in all directions. One might argue from this that it was not the expansion of gases by heat alone which caused the fractures. If these fractures were due to compressed gas, as contended by Jannettaz, one might expect this cracking to have occurred while the diamond and its contained gas were exposed to the enormous heat to which, according to the igneous theory, diamonds must have been exposed.

It is a strange fact that only light-brown or smoky stones

[20] *Bulletin de la Société Minerale de France*, ii., 1879, p. 124.

crack on being exposed to dry air at a slightly increased temperature. There are innumerable fragments of diamonds in the Kimberley pipes; and it is a question, how the original crystals became fractured.

6. Sir William Crookes says that the ash left after burning a diamond invariably contains iron as its chief constituent, and the most common colors of diamonds, when most perfectly pellucid, show various shades of brown and yellow from the palest "off-color" to almost black. These variations, he declares, accord with the theory that the diamond has separated from molten iron.

I have made exhaustive tests in order to ascertain whether diamonds contain iron, oxidized or metallic. The experiments were made with a magnetic separating-machine, the field-magnets of which attracted any mineral containing iron or iron oxides, except pyrites. Although some of these diamonds had the appearance of being coated with iron, and others were colored dark brown and deep yellow, they were in no way attracted by the magnet, even when excited by a strong electric current. These experiments do not, perhaps, disprove the existence of iron in the diamond, but they do establish the fact that the quantity is infinitesimally small. Further experiments in this direction ought to be made by those who have better facilities for such work than are at our disposal here in Kimberley. The experiments of Messrs. Hannay, Moissan, Friedel, Sir William Crookes, and others,—all show that microscopic diamonds can be produced artificially; but they throw very little light upon the question, how the diamonds in the South African craters crystallized.

7. From what is known of the theory of crystallization, one is inclined to the old Indian idea that diamonds grow like onions. It is hardly conceivable that diamonds, such as the Koh-i-nûr, the Great Mogul, the Excelsior (a Jagersfontein, South African stone of 971 carats), and the two De Beers diamonds (respectively of 503 and 428.5 carats), were formed, as the microscopic diamonds have been, in a moment of time during the sudden cooling of molten iron.

Is it not more reasonable to suppose that these enormous crystals grew little by little, and that nature has followed the same laws of crystallization in the diamond as in other minerals?

In March, 1904, a diamond of 228 carats was found in the Kimberley mine, which contains two red spots in the center of the stone. One of these is bright red, and, under a magnifying-glass, is shown to be a small diamond with crystalline faces easily distinguishable. The large diamond is cracked in all directions around the small crystal for a distance of about $\frac{1}{16}$ inch.

I regret that I am unable to propound a detailed theory as to the genesis of the diamond that could be supported by data less assailable and more conclusive than those given by others.

I have suggested difficulties which have occurred to me in reference to various theories, and which leave the subject still obscure.

All that can be said is, that carbon has been changed by crystallization from its black and uninviting appearance to the most beautiful gem which ever saw the light of day.

Centrifugal Ventilators.

BY R. V. NORRIS, WILKES-BARRE, PA.

(Lake Superior Meeting, September, 1904.)

In a former paper on this subject,[1] I summarized Murgue's theory and experiments, and presented a large number of tests of mine-fans, stating some conclusions, "more or less warranted by the evidence," as to design, capacity and efficiency. Since that time the very valuable report of Mr. M. Walton Brown,[2] Committee on Mechanical Ventilators, Institution of Mining Engineers, has appeared, and I have made additional experiments which tend to clear up some points in connection with this subject.

Design and Tests of the Luke-Fidler Colliery Fan.

The Luke-Fidler fan (see Figs. 1, 2 and 3), built to replace an old wooden fan at a very fiery colliery, presents some pecu-

[1] *Trans.*, xx., 637.

[2] *Transactions of the Institution of Mining Engineers* (London), vol. xvii., p. 482, and vol. xviii., p. 488.

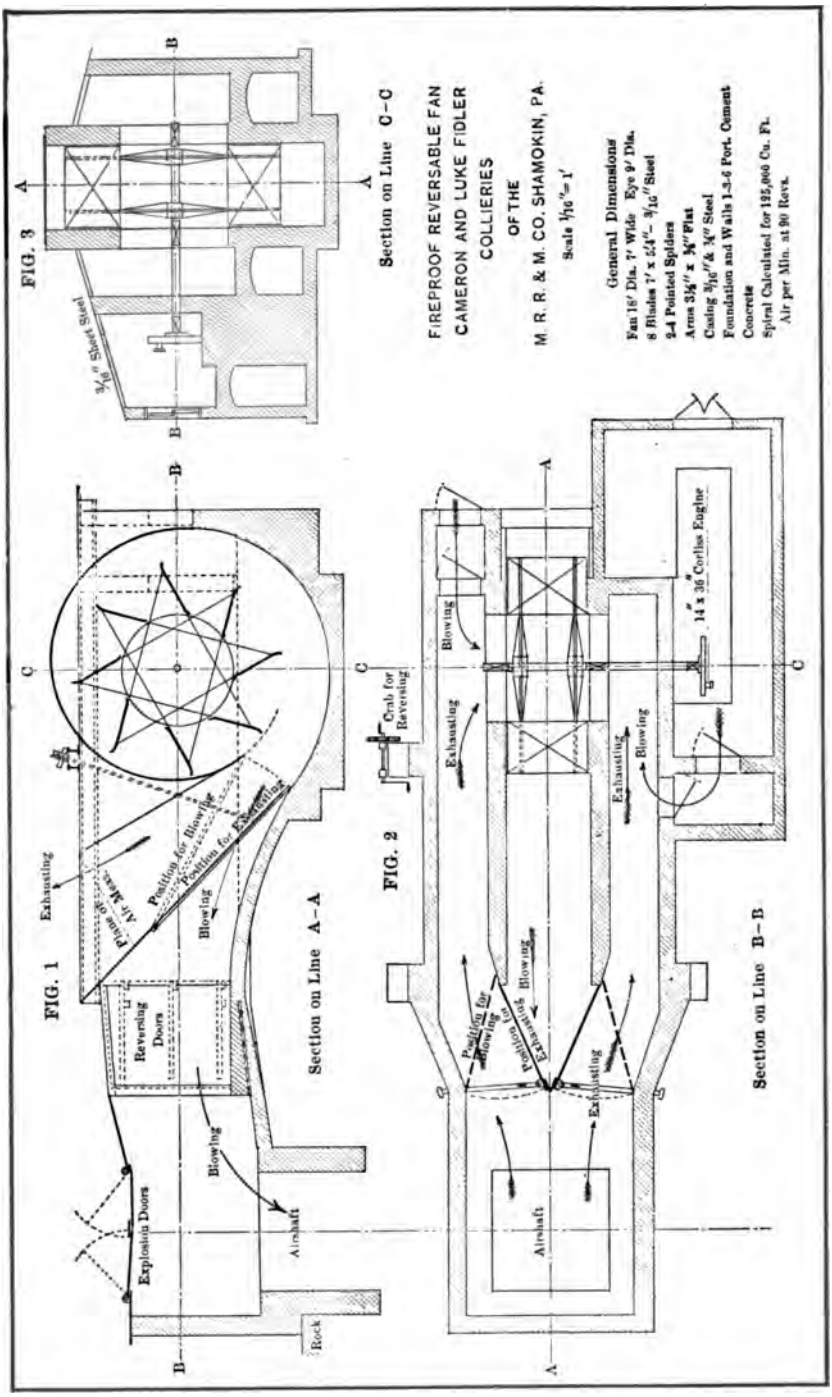


FIG. 1
Exhausting
Plane of Air Meas.
Position for Blowing
Position for Exhausting
Blowing
Reversing Doors
Blowing
Explosion Doors
Airshaft
Rock
Section on Line A-A
FIG. 2
Crab for Reversing
Blowing
Exhausting
Position for Blowing
Position for Exhausting
Blowing
Exhausting
Exhausting
Blowing
14" x 36" Corliss Engine
Airshaft
Exhausting
Section on Line B-B
FIG. 3
3/16" Sheet Steel
Section on Line C-C
FIREPROOF REVERSABLE FAN
CAMERON AND LUKE FIDLER
COLLIERIES
OF THE
M. R. R. & M. CO. SHAMOKIN, PA.
Scale 1/10"=1'
General Dimensions
Fan 16' Dia. 7' Wide Eye 9' Dia.
8 Blades 7' x 5'4"— 3/16" Steel
2-4 Pointed Spiders
Arms 3½" x ¾" Flat
Casing 3/16" & ¼" Steel
Foundation and Walls 1-3-6 Port. Cement Concrete
Spiral Calculated for 125,000 Cu. Ft. Air per Min. at 90 Revs.

liarities in design. Being constructed of concrete and iron, it is absolutely fire-proof; and, with the addition of the large explosion-doors over the air-shaft, it is hoped also explosion-proof. The construction of the fan permits the changing of the action in a few moments, from exhausting to blowing, by merely opening the double-reversing doors in the air-way and raising the movable sheet in the chimney (shown by the dotted lines in Figs. 1 and 2), both being done from the outside without necessarily stopping the fan or even altering its speed.

In order to facilitate the reversing and to improve the blowing-passages, the fan, originally designed as shown in Figs. 4 and 5, was turned 30° from the vertical, thus giving an inclined chimney-flue, shown in Fig. 1. In all other respects it was designed for an exhaust-fan which is its main work, the blowing feature being intended for use only in such emergencies, as mine-fires or explosions. There being a material difference in the design of an exhaust-fan and a blowing-fan, no compromise was attempted, and high efficiency as a blower was neither expected nor attained.

The general dimensions of the fan are as follows:—diameter, 18 ft.; width, 7 ft.; double intake, diameter of each eye, 9 ft.; 8 blades, 7 ft. by 5.4 ft., of $\frac{3}{16}$-in. steel, each stiffened by two 2.5 in. by 2.5 in. by 0.25-in. angles; flat arms, 3.5 in. by $\frac{5}{8}$ in. (as indicated in Fig. 1), two 4-pointed spiders, 4.5 ft. centers, all connected with fitted bolts in reamed holes; spiral casing, $\frac{1}{4}$-in. sheet-steel stiffened with 3-in. by 4-in. T-irons; casing concentric for 45° from the point of cut-off; balance of spiral calculated for 125,000 cu. ft. of air at 90 rev. per min., by the formula,

$$E = \frac{Q}{\pi DNB}.$$

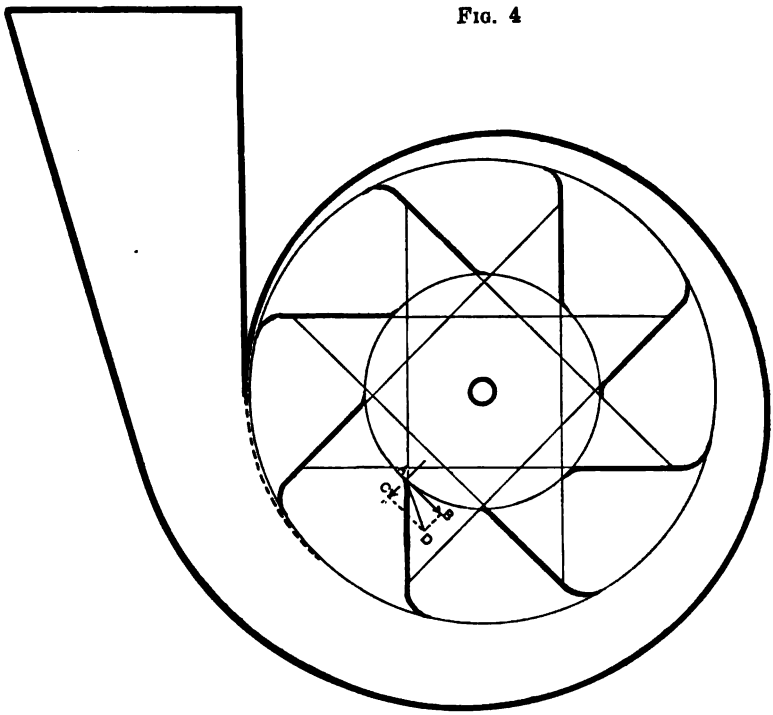
E = expansion at cut-off. Q = quantity in cu. ft. per min. D = diameter of fan in feet. N = rev. per min. B = breadth of fan-blade in feet.

The figures obtained by this formula give a velocity of air at the cut-off equal to that of the tips of the blades; and it has been my practice to add 20 per cent. to the value of E thus found, in order to allow for the slowing of the air in the casing.

The air-shaft, 10 ft. square, is lined with concrete to the rock, the explosion-doors directly over it open an area 11 ft. 8 in. by

12 ft.; the foundations, lower part of casing-walls, and floors of the building are of Portland-cement concrete, in the proportions of 1 of cement, 3 of sand, and 6 of broken stone, thoroughly mixed and well rammed in place. The roof is of $\frac{3}{16}$-in. steel supported on 3 by 4-in. T-iron, with cover-plates over all joints. The entrance-doors are of sheet-steel, $\frac{3}{16}$ in. thick, and the reversing-doors are 0.25 in. thick stiffened with angle-iron. The

FIG. 4



A. B.—Tangential speed inner edges of blades.
A. C.—Radial speed of air.
A. D.—Inclination of inner edges of blades.

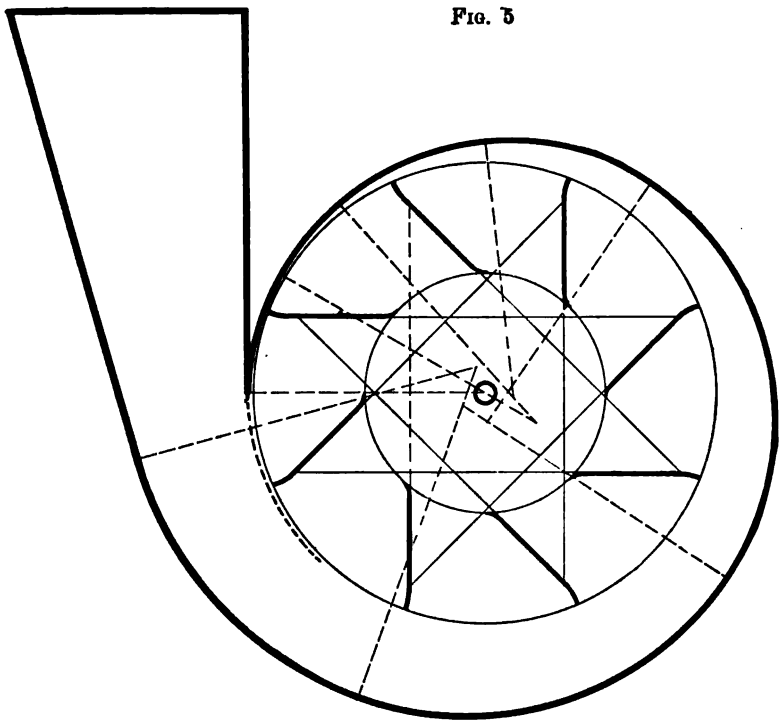
MODIFIED GUIBAL SPIRAL CASING, 18-FT. AMERICAN FAN, BLADE-TIPS BENT BACKWARD TO TANGENT.

engine is a 14 by 36 in. Corliss, direct-connected to the fan-shaft.

Shape of Blades.—For the purpose of determining the influence of the shape of the blade-tip on the efficiency of a fan, the tips were first bent backward to the tangent, shown in Fig. 4, and the fan was carefully tested. The blades were then changed by curving the tips forward to the radial shown in Fig. 5, and the tests were repeated.

Method of Testing.—The methods adopted in testing and the formulæ used in calculation were those used by the English Committee as detailed in its report,[3] except that, instead of changing the conditions in the mine, the air-shaft was tightly closed and air drawn in through the explosion-doors, the resistance being changed by varying the inlet-opening from the full opening of the explosion- and entrance-doors, down to an equivalent orifice of only 6 sq. ft.

FIG. 5



MODIFIED GUIBAL SPIRAL CASING, 18-FT. AMERICAN FAN, BLADE-TIPS BENT FORWARD TO RADIAL.

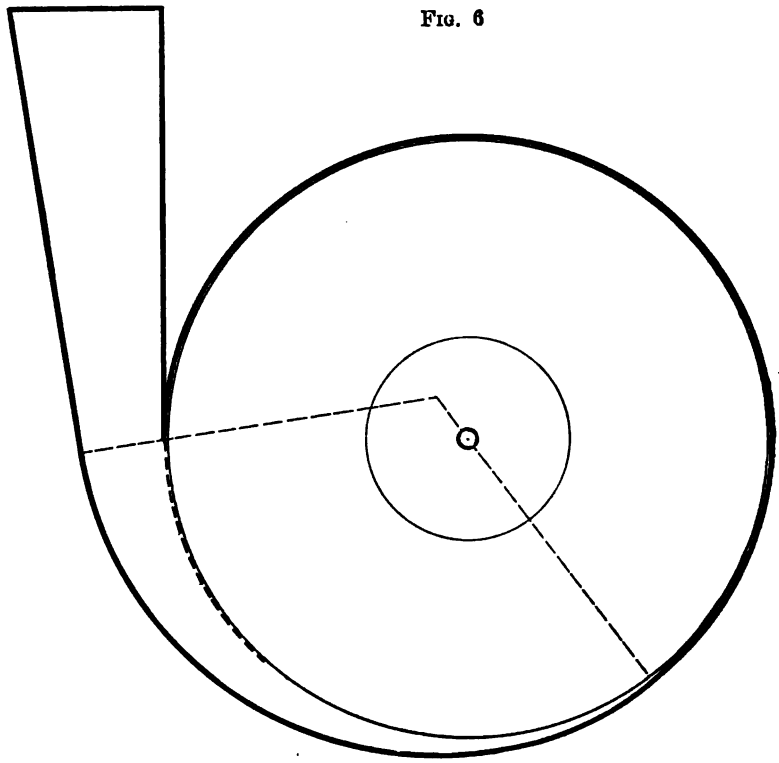
This method of testing has many advantages over any test made directly on the mine, for the reason that it avoids all complications and uncertainties due to natural ventilation. Any desired condition can be established almost instantaneously, maintained as long as desired, and re-established if necessary to verify results. The temperature and barometric corrections are simpler. Any fan can be tested under identical conditions,

[3] *Op cit.*

without regard to its particular mine; and there is no interference with the regular ventilation of the mine.

The water-gauge was taken close to the inlet by means of a pipe extending nearly to the outer end of the shaft, the end being protected by flannel so as to avoid variations due to the impact of the air. The air was necessarily measured in the chimney, at the point marked in Fig. 1, "plane of air-measurement." The area of the chimney at this point was divided by

FIG. 6



GUIBAL SPIRAL CASING OF A 36-Ft. ENGLISH FAN.

wires into 16 equal rectangles, and the anemometer, mounted on a rod, was held in the center of each rectangle for a given interval, usually one minute, timed with a stop-watch. The anemometer was then moved to the center of the next square, and so on until all had been covered, the total reading for the 16 squares being used in determining the average velocity of the air-current.

All the measurements were checked by the use of two ane-

TABLE I.—*Tests of* 18-*Ft. Fire-Proof Fan at the Luke-Fidler Colliery of the Mineral Railroad and Mining Co., Shamokin, Pa.*, 1903–04.

No. of Test.	Condition of Test.	Time.	Area of Office.	Corrected Anemom. Readings.	Area. Sq. Ft.	Cu. Ft. Air per Minute.	W. G. Center. Inches.	W. G. Drift. Inches.	Thermometer. Dry.	Thermometer. Wet.	Bar'r.	R. P. M.	Air per Rev. Cu. Ft.	Periphery Speed. Ft. per Minute.	Air per Min. at Normal Veloc. Cu. Ft.	W. G. at Norm. Vel. Inches.	Wt. of Air per Cu. Ft. Lbs.	Equivalent Orifice. Sq. Ft.	H.P. in Air Test.	I. H. P. of Engine Test.	Mech. Effic. Per Ct.	Indicator Cards. Nos.
	BLADE TIPS BENT BACK TO TANGENT.																					
1	Explosion-doors open. Nov. 10.	2.45	125.04	2794	59 62	166,578	0.625	0.5	61	50	29.18	85	1960	4807	176,400	0.70	0.0748	80.65	16.41	66.04	24.9	8-12
2	" " " " "	2.57	125.04	2822	59.62	168,248	0.625	0.5	60	50	29.18	88	1912	4976	172,080	0.66	0.0748	81.96	16.57	71.40	23.2	14-17
3	1/8 " " " " "	4.12	62.06	2537	59.62	151,256	0.875	.....	59	48	29.18	81	1867	4580	168,030	1.09	0 0744	61.60	20.86	56.87	36.6	18-21
4	1/4 " " " " "	4.42	29.98	1917	59.62	114,292	2.000	.....	60	48	29.18	96	1191	5429	107,190	1.76	0.0746	30.97	38.02	66.35	54.3	26-29
5	1/8 " " " " "	5.00	15.62	1152	59.62	68,682	2.000	.....	52	45	29.18	91	755	5146	67,950	1.96	0.0758	18.75	21.65	43.80	49.4	30-34
6	1/4 " " " " "	5.09	6.61	692	59.62	41,257	2.125	.....	58	46	29.18	90	458	5090	41,257	2.13	0.0750	10.86	13.81	36.16	38.2	35-39
7	1/4 " " " Nov. 11.	9.40	41.67	1711	59.62	102,010	1.063	.....	40	40	29 21	73	1398	4128	126,048	1.62	0.0775	38.64	17.8	39.8	44.7	44-45
8	1/8 " " " " "	9.45	41.67	2091	59.62	124,665	1.563	.....	40	40	29.21	90	1385	5090	124,665	1.57	0.0776	38.88	30.71	67.02	44.2	46-47
9	1/8 " " " " "	9.55	41.67	2227	59.62	132,774	1.938	.....	42	40	29.22	95	1398	5872	126,005	1.71	0.0774	37.22	40.6	77.7	52.2	48-49
10	1/8 " " " " "	10.05	61.64	2412	59.62	143,898	1.188	.....	43	41	29.22	90	1598	5090	143,908	1.19	0.0771	51.34	26.98	69.49	38.8	50-51
11	Explosion and fan doors open.	10.40	176.09	2908	59.62	173,375	0.5	.....	44	42	29.19	90	19.6	5090	173,375	0.50	0.0767	95.26	13.66	78.37	17.4	55-57
	BLADE TIPS BENT FORWARD TO RADIAL.																					
12	Fan closed. Jan. 4.	2.00	.....	.....	.....	.....	2.2	2.1	12	.....	29.7	88	.....	4976	.....	2.31	0.0835	.....	.....	31.8	0	1-2
13	One explosion-door open.	2.10	57.0	3045	59.75	182,569	1.4	1.2	13	.....	29.7	89½	2044	5061	184,340	1.43	0.0834	62 39	40.3	101.7	39.6	3-4-5
14	" " " "	2.27	57.0	2934	59.75	177,018	1.54	1.2	14	.....	29.7	90⅝	1958	5110	176,672	1.53	0.0832	57.63	43.0	99.1	43.4	6-7-8
15	" " "17" "	2.45	24.0	1462	59.75	88,207	2.3	2.3	14	.....	29.7	90¼	977	5107	88,081	2.29	0.0832	28.50	31.9	65.5	48.7	11-12
16	" " " 8½" "	3.02	12.0	963	59.75	58.101	2.51	2.4	16	.....	29.7	91¾	632	5188	57,115	2.32	0.0828	15.13	23.0	48.3	47.6	13-14
17	" " " 4¼" "	3.13	6.0	523	59.75	31,675	2.56	2.5	16	.....	29.7	92	344	5203	31.048	2.46	0.0828	7.98	12.8	39.3	32.6	15-16
18	" " "24" "	3.25	30.0	2376	59.75	144,076	2.56	2.4	16	.....	29.7	93	1501	5429	135,345	2.26	0.0828	36.30	58.1	94.5	61.5	17-18
19	" " "24" "	3.35	30.0	2364	59.75	142,628	2.56	2.2	14	.....	29.7	96	1486	5429	133,985	2.26	0.0832	36.02	57.5	95.5	60.2	19-20
20	Explosion-doors full open.	3 56	110.0	3274	59.75	197,531	0.92	0.7	13	.....	29 7	88	2245	4976	202,454	0.97	0.0834	83.29	28.6	94.5	30.3	23-24
21	" " " "	4.13	110.0	2605	59.75	157,174	0.57	0.4	12	.....	29.7	71¾	2175	4060	197,435	0.90	0.0835	84.28	14.1	55.1	25.6	25-26
22	" " " "	4.20	110.0	1617	59.75	97,559	0.25	0.8	12	.....	29.7	46	2121	2601	191,292	0.96	0.0835	79.00	8.8	17.0	22.6	27-28-29
23	" " closed. Jan. 5.	10.50	Mine.	836	59.75	50,411	2.15	.....	-18	.....	29.7	89	567	5033	51,082	2.21	0.0902	14.44	17.1	37.5	45.6	38-39-40

* Normal velocity—5,100 ft. per min.

mometers, either simultaneously, beginning at opposite sides, or by repeating the measurements. Both anemometers were tested and the proper corrections made to the readings obtained. The details and results of these tests are given in Table I., and

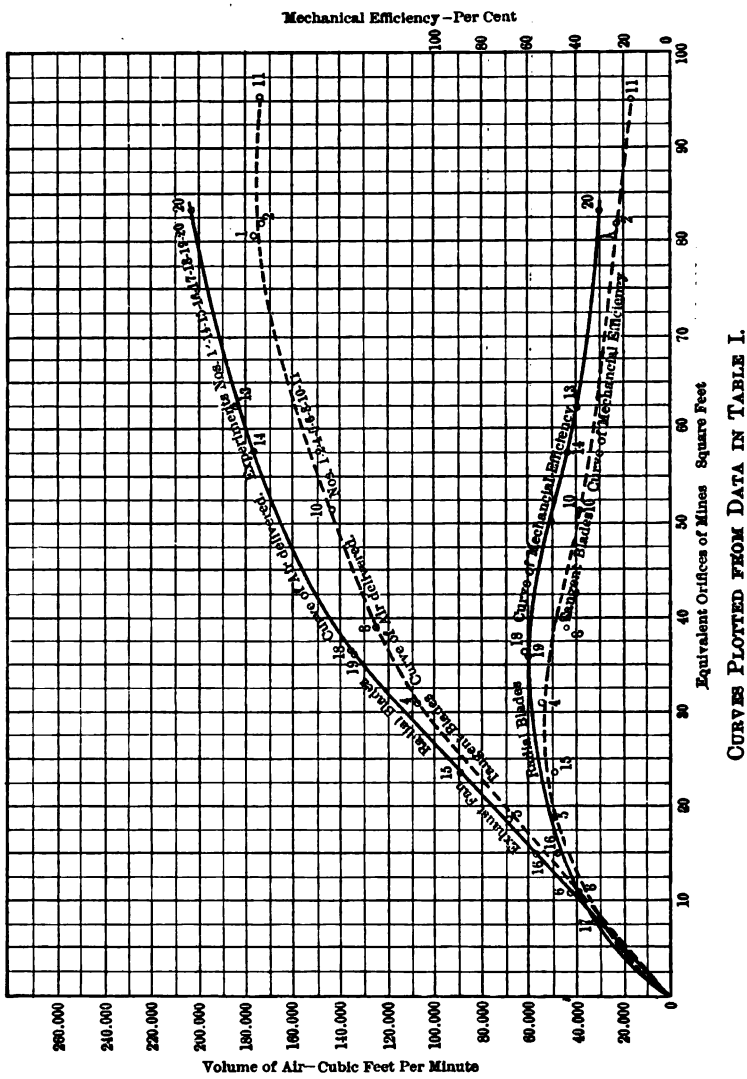


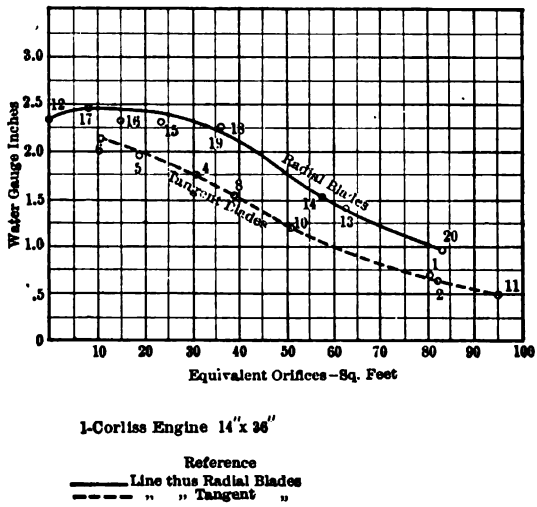
FIG. 7.—CURVES PLOTTED FROM DATA IN TABLE I.

the corresponding diagrams, showing the curves of air delivered, the mechanical efficiencies, and the water-gauges for varying orifices, with blade-tips tangential and radial, are given in Figs. 7 and 8. From the data in Table I., it is apparent that

the radial-tip blades have a very decided advantage in quantity of air delivered, in the water-gauge maintained and in efficiency. These results agree with Mr. Murgue's theoretical investigation. The tangent-tip, however, is a sufficiently common practice in the anthracite region to warrant the experimental investigation.

The diagrams show clearly the proper capacity of this particular fan, and it is interesting to note that the point of highest efficiency occurs about at the point where the curve of air delivered (called the characteristic curve of the fan by the Com-

FIG. 8.



CURVES PLOTTED FROM DATA IN TABLE I.

mittee on Mechanical Ventilators) begins to spring downwards from a straight line.

COMPARISON OF RESULTS.

In order to ascertain whether this result was accidental, or was characteristic of Guibal fans, I have plotted the curves of other fans in the diagram, Fig. 9, using the tests of the Committee on Mechanical Ventilators as well as a few from my former paper, adding also the curves of the Scheile, Waddle, and Capell fans from the same sources. This diagram shows that each size and type of fan has its own characteristic curve, but that all fans of the same type, on mines of the same resist-

TABLE II.—*Comparative Tests of Mine Fans on Mines of Varying Resistance.*

Reference Number.	Description of Fan.	Anemometer Readings. Corrected Feet.	Area of Measurement. Sq. Ft.	Cu. Ft. Air per Minute.	Water Gauge. Center Inches.	Water Gauge. Drift Inches.	Thermometer. Dry.	Thermometer. Wet.	Bar. Inches.	Rev's per Minute. Fan.	Air per Revolution. Cu. Ft.	Periphery Speed. Feet per Min.	Air per Min. of 5,100 Periphery Speed. Cu. Ft.	W. G. of 5,100 Periphery Speed. Inches.	Avg. Weight of Air per Cu. Ft. Lbs.	Equivalent Orifice of Mine. Sq. Ft.	H.P. in Air-Test.	H.P. of Engine-Test.	Mechanical Efficiency. Per Cent.	
1	Great Western Colliery:	23½	102.37	2,406	0.05	0.00	63½	61¾	29.185	6¾	356	848	14,086	1.807	0.0737	4.085	0.019	2.86	0.70	
2	Guibal fan, 40 ft. dia., 12 ft. wide;	376¾	"	38,517	2.59	—0.02	66¾	64	29.185	47⅞	805	6,016	32.643	1.860	0.0734	9.091	15.704	98.55	15.93	
3	dia. single intake, 14 ft.	526⅓	"	53,898	2.57	—0.02	67	64½	29.185	48	1,123	6,032	45,565	1.836	0.0733	12.776	21.806	102.84	21.20	
4	Tested June 2, 1889.	2,055½	"	210,396	2.44	1.85	64½	63½	29.18	46⅝	4,513	5,850	183,113	1.848	0.0735	51.301	80.86	146.12	55.32	
5		3,309¾	"	338,819	1.55	0.42	62½	60½	29.20	41⅜	8,189	5,199	332,300	1.490	0.0738	103.898	82.69	161.72	51.13	
6	Pelton Colliery:	359½	78.53	28,232	2.86	2.82	56	56	29.71	54⅛	522	6,121	23,520	1.985	0.0770	6.05	12.72	68.87	18.47	
7	Guibal fan, 36 ft. dia., 12 ft. wide;	546½	"	42,917	2.85	2.80	56½	56½	29.72	53¼	806	6,022	36,342	2.043	0.0770	9.89	19.27	68.60	28.09	
8	dia. single intake, 13 ft.	2,177	"	170,960	3.65	3.68	61	61	29.82	61¼	2,791	6,927	127,861	1.978	0.0767	34.81	98.34	174.95	56.21	
9	Test, January 19 and 20, 1889.	2,243	"	176,104	2.57	2.53	59¼	59¼	29.82	51⅜	3,428	5,810	154,568	1.960	0.0768	42.75	71.33	123.48	57.76	*a*
10		3,241	"	254,516	1.98	1.88	45½	45½	29.81	51⅞	4,906	5,868	221,338	1.494	0.0779	70.79	79.34	184.36	43.03	
11	Wingate Grange Colliery:	284½	89.40	25,150	1.60	1.53	91½	91	29.46	40⅛	621	4,580	28,000	1.977	0.0745	7.66	6.32	28.10	22.49	
12	Guibal fan, 36 ft. dia., 12 ft. wide;	707½	"	62,543	1.54	1.52	92	91½	29.45	39⅝	1,578	4,481	71,172	1.989	0.0743	19.28	15.14	28.65	52.83	
13	dia. single intake, 13 ft.	1,658⅔	"	146,611	2.64	2.34	80½	79½	29.60	52¼	2,806	5,909	126,517	1.966	0.0751	34.68	61.01	110.46	55.23	
14	Test, February 2 and 3, 1889.	1,547	"	136,755	1.61	1.44	79½	79	29.65	41¼	3,315	4,665	149,494	1.928	0.0754	41.47	34.79	67.57	51.48	
15		1,036⅓	"	93,395	1.61	0.54	80½	79½	29.20	25⅝	3,610	2,926	162,760	1.852	0.0742	45.91	8.98	17.66	50.88	
16		2,041⅔	"	180,469	1.12	0.99	79½	79	29.66	37⅜	4,766	4,283	214,859	1.593	0.0758	65.50	31.94	69.08	46.23	
17	Guibal fan, 25′ dia., 8′ wide, 11′ 6″ intake.	.....	.....	137,760	.....	2.55	.....	.....	.....	70	1,825	5,495	127,860	2.19	.....	32 0	55.35	67.44	82.07	No. 3 B
18	" " 25′ " 8′ " 11′ 6″ "	.....	.....	225,288	.....	1.45	.....	.....	.....	62	3,634	4,867	236,100	1.59	.....	75.0	51.44	62.93	81.74	" 4
19	" " 24′ " 10′ " 12′ 0″ "	.....	.....	194,200	.....	2.50	.....	.....	.....	73	2,660	5,504	179,950	2.14	.....	44.0	75.69	101.07	75.69	" 10 B
20	" " 25′ " 8′ " 12′ 0″ "	.....	.....	203,280	.....	1.80	.....	.....	.....	70	2,904	5,495	188,670	1.55	.....	52.0	57.70	107.10	54.00	" 13 I
21	" " 35′ " 11′ 8‴ " 17′ 6″ "	.....	.....	228,850	.....	1.95	.....	.....	.....	46	4,975	5,060	230,660	1.98	.....	66.0	70.62	98.45	71.73	" 8
22	" " 35′ " 11′ 8‴ " 17′ 6″ "	.....	.....	234,830	.....	1.85	.....	.....	.....	46	4,648	5,060	236,690	1.88	.....	68.0	68.46	94.60	72.37	" 9 A
23	" " 35′ " 11′ 8‴ " 17′ 6″ "	.....	.....	257,520	.....	2.40	.....	.....	.....	50	5,150	5,500	238,790	2.06	.....	66.5	97.40	131.30	74.28	" 11⅛ B
24	" " 35′ " 12′ 0‴ " 17′ 6″ "	.....	.....	177,000	.....	2.25	.....	.....	.....	48	3,687	5,278	171,080	2.10	.....	47.0	62.70	82.20	76.20	" 14 A

b

a *Transactions of the Institution of Mining Engineers*, "Mechanical Ventilators," M. Walton Brown, 1900.

b *Trans.*, xx., 642, "Centrifugal Ventilators," R. V. Norris, 1891.

Equivalent orifice $= \frac{v}{90.824\sqrt{h\frac{d_g}{d_g}}}$ (a. p. 20)

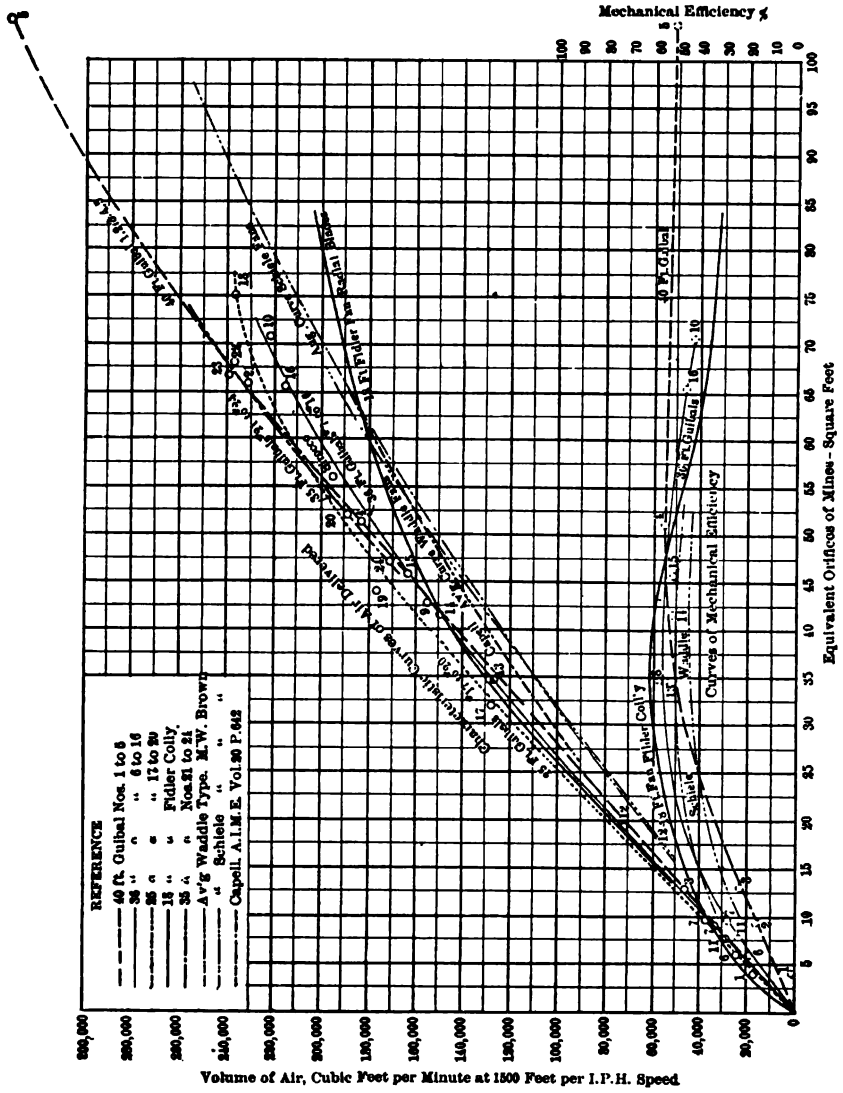
v = observed volume of air cu. ft. per min.
h = observed water gauge.
d_g = mean wt. per cu. ft. water.
d_g = " " " " " air.

Equivalent orifice $= \frac{0.408Q}{\sqrt{h}}$ Murgue, p. vii.

Q = Quantity air in 1,000's cu. ft. per min.

ance, give practically the same quantity of air up to the capacity of each fan, when the peripheral speed of tips of blades is the same. Up to 45 sq. ft. equivalent orifice the curves of all the

FIG. 9.



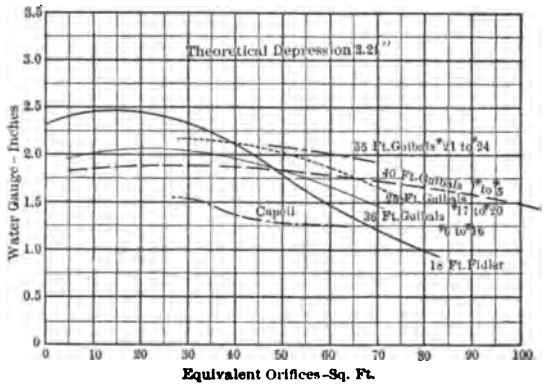
CURVES PLOTTED FROM DATA OF COMMITTEE ON MECHANICAL VENTILATORS.

Guibal fans practically coincide; at this point the curve of the 18-ft. fan begins to deflect downwards, and, as the equivalent orifice is enlarged, the curves of the larger fans likewise drop.

The curves of the other three types of fans follow the same law, but unfortunately the available tests on varying sizes are too few to extend the curves in detail.

By comparing the efficiencies with the characteristic curves, three results are noteworthy, namely,—1. Each size of fan reaches its best mechanical efficiency at about the point where its curve begins to vary from a straight line. 2. Each size of fan is at its best on mines of a certain equivalent orifice which for the highest efficiency can vary only within rather narrow limits. 3. The mechanical efficiency is poor, with fans too large for their mines as well as for those too small. These results show conclusively the absolute necessity of designing a fan

FIG. 10.



CURVES OF WATER GAUGE.

with reference to its proposed work, and the futility of attempting to use a standard size or design of fan for different mines.

The curves of water gauge (see Fig. 10), also, show that the depression of the same fan varies with different resistances, and illustrates the uselessness of the old term "manometric efficiency" which, in the case of the Fidler fan, varies from 73 per cent. at 8 sq. ft. to only 30 per cent. at 83 sq. ft. equivalent orifice. It is noticeable, however, that the fans with large spirals maintain a higher depression up to their capacity than do the English fans, and that the depression-curves flatten and hold up better as the diameter of the fan increases.

It is also noticeable that the curves of each size of American fans lie above those of similar sizes of English ones. This re-

sult is probably due to two causes: (1) the shape of the casing, the English being constructed on Guibal's original lines, Fig. 6, with a closed casing for about two-thirds of the circumference and a contracted outlet; and the American fans, Figs. 4 and 5, with spirals starting near the cut-off and expanding all around to the chimney; and (2) the great variation in size of eye as compared with the diameter of the fan, the English fans having eyes rather less than one-third the diameter, while those of the American fans are at least one-half.

It is probably unfair to the Capell fan to include these old tests in the diagrams, especially as it is essentially a high-speed fan, and most of the tests available had to be reduced from periphery-speeds of 7,000 ft. and upwards per min., but, as a request to the American agent of this fan for further data has met with no response, the old figures are given for what they are worth. It is interesting, however, to note that its characteristic curve, when reduced to the same speed, falls between those of the Guibal, and those of the Schiele and Waddle types.

The only available test of the new Sirocco fan is one made on a mine having 56 sq. ft. equivalent orifice, which shows that this 6.25-ft. fan gave about the same quantity of air as the larger Guibal fans; unfortunately, no figures of efficiency of the Sirocco fan are available.

In view of the further data now given, the unsatisfactory conclusions of my former paper[4] should be revised; and the following are presented in their place.

1. *Influence of Resistance of Mine (Equivalent Orifice) on Fan.*—The quantity of air delivered by any fan at a given speed is practically proportional to the equivalent orifice of the mine, up to the property capacity of the fan; beyond this capacity the proportion rapidly decreases, as shown by the characteristic curves, tests 1 and 2.

The highest mechanical efficiency of any fan is reached with a mine which permits it to work at or near its proper capacity.

2. *Influence of the Diameter of a Fan on Its Performance.*—The proper capacity of a fan is largely dependent upon its diameter, hence the size should be determined with reference to the resistance of the mine upon which it is to work; the efficiency

[4] *Trans.*, xx., 665.

of any fan depending largely upon its working at or near its proper capacity.

3. *Influence of the Width of a Fan on Its Performance.*—Very little experimental data is available on this point, but it seems reasonable to believe that the cylindrical inner surface of the blades should be approximately equal to the clear area of the inlet eye or eyes. The fan, 20 ft. in diameter and 12 ft. wide, referred to in my earlier paper,[5] was tried and proved a comparative failure; both the quantity of air delivered and the efficiency were low, due probably to its excessive width in comparison with its area of intake.

4. *Influence of Shape of Blades of a Fan on Its Performance.*—The conclusive tests of the Fidler fan, Fig. 8, prove that the tips of the blades of a Guibal fan should be at least radial. No experiments are available, however, on blades bent forward beyond this point, though it seems reasonable to believe that the radial tips are correct. The gain in the quantity of air delivered, in mechanical efficiency, and in water-gauge, accomplished by changing the blade-tips from tangential to radial position, is too great to have been the result of any possible error in experiment, especially as the tests were made by the same observers and apparatus upon the same fan, changed only in this particular, and free from any influence of the mine.

The inner edges of the blades should have such an inclination as to receive the air without shock. This effect seems to be attained by making the angle between the inside of blades and a tangent to the inner surface the resultant of the radial speed of the air at this point, and the tangential speed of the inner edges of the blades.

5. *Influence of the Shape of the Spiral Casing of a Fan on Its Performance.*—This influence is considerable, the large casing, Figs. 4 and 5, not only giving higher efficiencies than the close casing, Fig. 6, but higher water-gauge and larger volumes of air under the same conditions; the American Guibal 25-ft. fan being nearly equal, in the quantity of air, to the English 36-ft., and the American 35-ft. fan to the English 40-ft. The American fans are also superior in the water-gauge maintained.

[5] *Op cit.*

The spiral casing for any fan should be designed with reference to its expected work.

6. *Influence of the Diameter of Inlet Eye of a Fan on Its Performance.*—This influence may be considerable, and with the casing it may be taken as a reason for the superiority of the American type. Experimental data on this subject are woefully lacking, but it seems probable that the ratio of the diameter of inlet to the diameter of fan should vary with the proposed work. Fans with large inlets may probably prove suitable for large quantities of air and low-water gauges, and those with relatively small inlets for the reverse conditions. Relatively large inlets would certainly decrease the internal resistance of a fan and thereby increase its proper capacity.

7. *Influence of the Speed of a Fan upon Its Performance.*—An analysis of many experiments shows, that while the air delivered by any fan is practically directly proportional to its speed and that the water-gauge given with the same equivalent orifice is proportional to the square of the speed, the mechanical efficiency is much affected by the speed at which a fan is run. Tests Nos. 20, 21 and 22 of the Fidler fan, and No. 13 A. to 13 J. of the South West Coal and Coke Co. fan,[6] show clearly the variation in efficiency due to speed, and also verify experimentally the laws relating to air-delivery and water-gauge.

The question of the proper speed for each type and size of fan requires much additional experiment for its proper determination, though it still is probable that for Guibal fans, at least, a peripheral speed of from 5,000 to 6,000 ft. per min. is the best, the proper size of fan being adopted to obtain the required air at about that speed, and more than one fan used if necessary for this purpose.

It is hoped that this present paper may awaken interest in this much-neglected subject, as only by the co-operation of many engineers can sufficient data be accumulated to settle the many vexed questions pertaining to the design of a ventilating-fan. This desired result may best be accomplished by the publication of the characteristic, efficiency and water-gauge curves of various sizes and designs of fans, so that, from the whole, it may be possible, eventually, to select a fan best suited to the particular conditions under which it is required to operate.

[6] *Trans.*, xx., 642.

The Manufacture of Coke in Peru.

BY J. MORGAN CLEMENTS, NEW YORK CITY.

(Lake Superior Meeting, September, 1904.)

THE manufacture of coke in Peru, as practiced at the coal-mines of the Quishuarcancha and Goyllarisquisca districts, is intermediate between the primitive coke-heap and the bee-hive oven.

The method of coking is quite simple. In constructing the coal-heap, preparatory to burning, the ground is first leveled, and a simple stack of stone and clay is built, giving the bottom, to a height of 6 ft., approximately 6 ft. of external diameter. The size of the stack varies in different districts, but is generally 9 ft. high, and slightly smaller at the top than at the bottom. The walls of the heap are from 12 to 14 in. thick, which allows an inside diameter of from 3.75 to 4 ft. These walls are pierced at intervals by three or more tiers of openings or flues, connected with the central flue. The radial flues have a diameter of about 5 in. and are built with a gentle upward slope. The stack (shown in Fig. 1) is permanent; and in this respect the practice in Peru differs from other primitive practices in which nothing is permanent except the yard.

Above the permanent 6-ft. portion of the stack there is added a cap of clay, about 3 ft. high, which contracts toward the top, where it has an inside diameter of about 2 ft. This clay cap is temporary, and frequently has to be renewed after each burning.

Before charging the coal, there are first built, from the lowest tier of flues upon the ground, a number of temporary flues, leading from the central stack out to the edge of the coal-heap, and constructed of blocks of coal for the greater part of the distance, the outer edge alone being of rough stone. The coal is then piled up over these flues and around the stack to a height of about 6 ft. The bottom flues are the only ones that

FIG. 1.

PERMANENT CENTRAL STACK OF COKE HEAP, SHOWING OPENINGS INTO FLUE, QUISHUARCANCHA, PERU.

FIG. 2.

COKE-HEAPS AT QUISHUARCANCHA, PERU.

extend to the periphery. The heap, about 20 ft. in diameter, contains about 24 short tons of coal.

The coal is ignited by means of small fires built at the ends of the flues at the base of the heap, and from these the combustion gradually extends throughout the entire heap. About 10 days are required for coking; and 12 tons of a fair grade of coke are obtained. A good deal of ash accumulates upon the top of the pile and around the flues resulting from the burning of the coal with free access of air. In some cases the practice is modified slightly by spreading a layer of sod over the heap, which partially excludes the air from the surface. The columns of coke are vertical or generally normal to the surface of the heap, and from 12 to 18 in. long. The coke produced in this crude way has a silvery luster, good structure, and is fairly strong; but, by reason of the method of burning, as well as the character of the coal used, the percentage of ash is high.

A view of the coke-heaps at Quishuarcancha is shown in Fig. 2. The heap to the left of the view is nearly ready to be torn apart, and the temporary cap of this stack has fallen down. The cap on the stack to the right is still present.

The Geology of the Treadwell Ore-Deposits,* Douglas Island, Alaska.

BY ARTHUR C. SPENCER, WASHINGTON, D. C.

(Lake Superior Meeting, October, 1904.)

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I. INTRODUCTION.

THE object of the following paper is a description of the Treadwell gold-deposits in their geological aspects, quite apart from any consideration of the economical methods of mining and milling which have been developed and successfully applied

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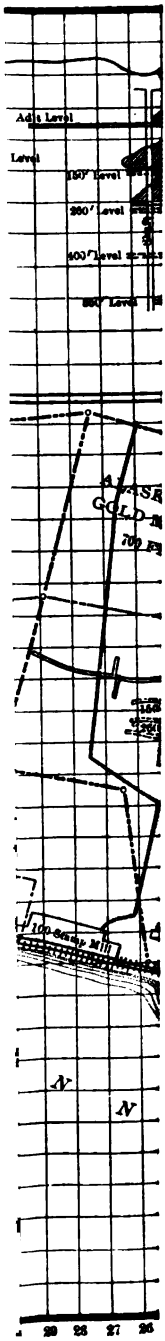
to the working of ores having an average value of little over $2 per ton. The studies upon which it is based were made in course of a general official survey of the Juneau gold-belt during the summer of 1903, under the instructions of Alfred H. Brooks, Geologist in Charge of the Division of Alaskan Mineral Resources, United States Geological Survey.

To the Hon. William Alvord, President of the Alaska-Treadwell Gold Mining Co., to Mr. Joseph McDonald, Superintendent of the mines, and to Mr. Robert A. Kinzie, Assistant Superintendent, I am greatly indebted for their hearty co-operation and uniform courtesy during my examination of the Douglas Island mines.

Douglas island, upon which these renowned mines are situated, is one of the smaller islands of the Alexander archipelago, separated from the mainland of southeastern Alaska by a narrow fiord known as Gastineau channel. On the island two towns, Douglas and Treadwell, owe their 2,000 inhabitants to the activity of the mining operations, while Juneau, with its somewhat greater population, lies on the adjacent mainland, about 2.5 miles northwest of Treadwell. The distance by steamer from Seattle, Wash., to Juneau slightly exceeds 900 miles, while Skagway, the terminus of the Yukon & White Pass railroad, lies 95 miles to the northwest, and Sitka about the same distance in a southwesterly direction.

The four mines of the Treadwell group are located near the inland shore of Douglas island. From the Ready Bullion mine (Fig. 1), which is the most southeasterly one, it is 3,000 ft. to the nearest workings of the Alaska-Mexican mine, and the intervening ground is supposed to be practically barren. The Mexican workings extend, however, almost to those of the Seven Hundred Foot property, and the latter connect at several levels with the tunnels of the Alaska-Treadwell mine, showing an almost continuously developed ore-body for a distance of about 3,500 feet. Although the workings have revealed several separate ore-bodies, and certain distinctions are made in the character and occurrence of the ores, the mines are all located on the same lead, and the ore-material is practically of one nature and of identical origin throughout. As a whole, therefore, the deposits may be conveniently designated under the name of the first discovered and largest mine.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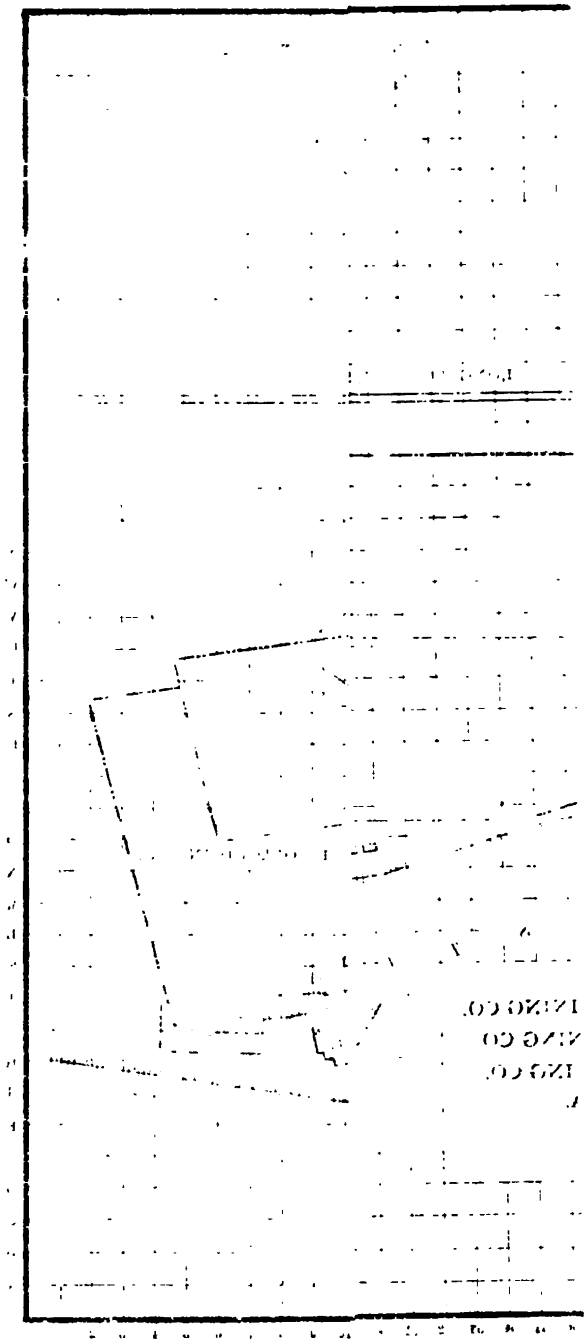
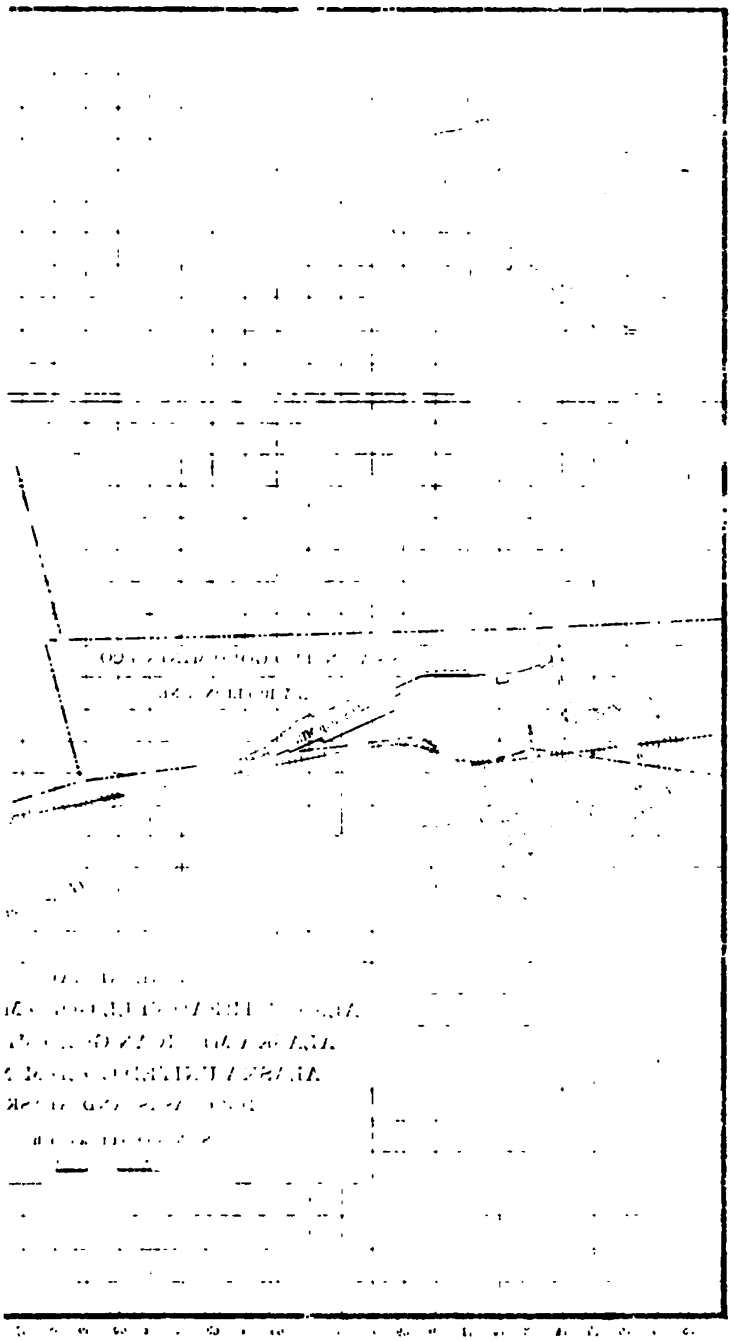




Descriptions of the mines, upon which the present general knowledge of their geological features rest, have been published by Dr. Geo. M. Dawson[1] and by Dr. Geo. F. Becker,[2] while the microscopical character of the ore-material is described in detail by Dr. F. D. Adams,[3] and some notes have been recently contributed by Dr. Charles Palache.[4]

A concise account of the mines, from a commercial and engineering standpoint, is given by J. H. Curle,[5] and recently the methods and statistics of mining and milling have been treated at length by Robert A. Kinzie,[6] the present assistant superintendent of the mines.

Dr. Dawson's cursory examination was made in 1887, before extensive openings had been made; and while Dr. Becker had better opportunity for observation in 1895, the workings were still rather limited at that time. During the eight years which have intervened since the visit of the latter, the developments have been very extensive, and the present facilities for investigation are all that could be desired. Additional facts have therefore been secured, and it has been possible to bring to the study of the problems presented by the deposits, a fairly accurate knowledge of the general geology of the Juneau region. Under these circumstances disagreement with some of the suggestions and conclusions of the earlier investigators is naturally to be expected, but such differences as have appeared are mainly in regard to the order of geological events, and have no practical bearing upon the economic future of the deposits.

Before proceeding to the detailed descriptions, a brief outline of the general geological features of the region will be presented.

II. Geology of the Region.[7]

General Features.—Southeastern Alaska, sometimes called the Panhandle, is the portion of the Territory lying between the

[1] *American Geologist*, vol. iv., 1889, pp. 84–88.

[2] *18th Annual Report U. S. Geological Survey*, Pt. III., pp. 7–86.

[3] *American Geologist*, vol. iv., 1889, pp. 88–93.

[4] *Alaska. Harriman Alaska Expedition*, vol. iv., Geology and Paleontology. New York, 1904, pp. 59–66.

[5] *Gold Mines of the World*, 2d Edition. New York and London, 1902.

[6] *Trans.*, xxxiv., 334–386.

[7] The Juneau Gold-Belt, by Arthur C. Spencer. *Bulletin* 225, *U. S. Geological Survey*, Washington, 1904, pp. 28–42.

Pacific ocean and British Columbia. It comprises the islands of the Alexander archipelago and a mainland-strip about 30 miles wide. In this relatively narrow mountainous belt the geological formations are disposed in bands parallel with the general northwest trend of the mainland coast and the longer dimensions of the many islands which dot the archipelago. On the mainland three easily distinguishable groups of rocks can be traced from Windham bay on the south to Berners bay, north of Juneau, a distance of nearly 150 miles.

The main mass of the mountains, between the International Boundary and the innermost salt-water passages followed by the coastwise-steamers, is formed by intrusive rocks related to diorite. In front or to the southwest of this formation there is a band of metamorphic schists, followed by a zone of alternating black slates and greenstones, which together constitute the principal bed-rock of a coastal strip from 3 to 8 miles in width. These metamorphosed sedimentary rocks have been correlated in a general way by Brooks[8] with the Ketchikan schists studied by him in the southern part of the archipelago, and these in turn correspond in part to the Vancouver series of Dawson. However, the Vancouver series was known to include both Carboniferous and Triassic strata, and Dawson regarded the slates occurring near Juneau as equivalent to the Triassic part of the series. Fossils discovered during the field-season of 1903 now show that at least part of the stratified rocks of the Juneau region are of Paleozoic age.

The strike of the stratification, and of the almost omnipresent secondary schistosity, and also the courses of nearly all intrusive contacts, vary between N. 20° W. and N. 40° W. The prevailing dips are away from the sea, that is, toward the northeast; the inclination of the rocks varying as a rule from 20° to 70°, but with local instances of vertical or slightly overturned position.

The attitude of the formations occurring in this mainland portion of southeastern Alaska points to simple structure like monoclinal tilting, and though this would require a very thick succession of sedimentary rocks, no duplication of beds, either

[8] The Ketchikan Mining District, Alaska, by Alfred H. Brooks. *U. S. Geological Survey, Professional Paper, No.* 1, Washington, 1902.

upon a small or grand scale, has yet been discovered to suggest the existence of a system of isoclinal folds or of a series of

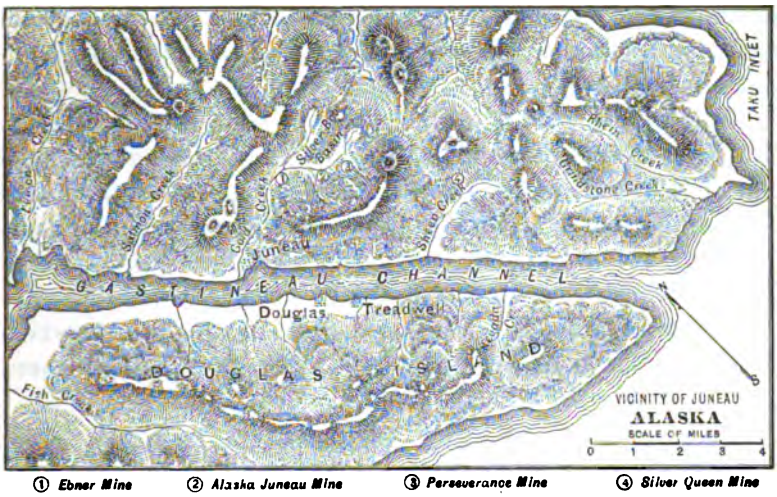


FIG. 2.—TOPOGRAPHY NEAR JUNEAU, ALASKA.

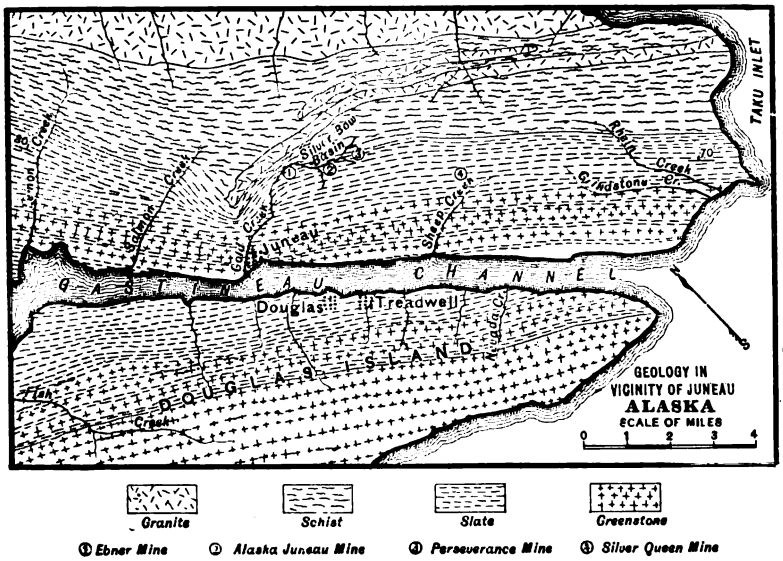


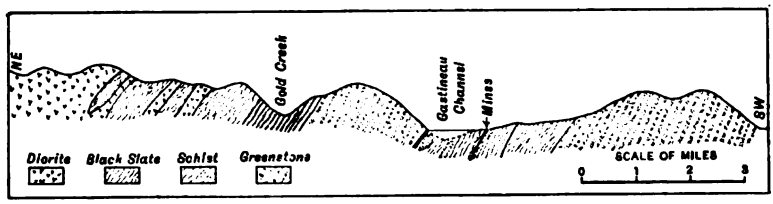
FIG. 3.—GEOLOGY NEAR JUNEAU, ALASKA (CORRESPONDING TO FIG. 2).

block faults, either of which would greatly reduce the apparent thickness.

The general character of the topography near Juneau is shown in the sketch map, Fig. 2. The distribution of the formations in the same area is given in Fig. 3, while the cross-section (Fig. 4) drawn through Douglas island and the adjacent mainland illustrates the general structural relations of the formations.

The geology of the Juneau region and of southeastern Alaska as a whole resembles, in many ways, that of the gold-belt of California. The rocks of both regions are in large part of identical character, and some of them correspond in age and in the nature of their metamorphism. There is also a marked similarity in the occurrence of the gold-veins and in the general effects of mineralization, and some of the broader facts suggest that the dates of vein- and ore-deposition also corre-

FIG. 4.



GEOLOGICAL SECTION OF DOUGLAS ISLAND AND MAINLAND NEAR JUNEAU, ALASKA.

spond closely, though more extended and further detailed studies must be made before definite proof of this can be obtained.

Intrusive Rocks of the Coast Range.—The main mass of the mountainous belt from 50 to 80 miles wide, which separates the waters of the Pacific from the plateau region of British Columbia, is composed of coarse-grained intrusive rocks which have been designated the Coast Range granites by Dr. Dawson, of the Canadian Geological Survey. In the Juneau belt true granite rarely occurs, and while the black and white granular rocks of the region often pass under this designation and are undoubtedly part of, or closely connected with, the intrusives of the whole Coast range; as a rule they are really diorites or related rocks, many of them being similar in composition and appearance to the diorites and granodiorites of the Sierra Nevada in California.

Within the diorite-zone of the Coast range there are various

minor bands of included metamorphic rocks similar to those which form its bounding formations, and there are also some basic dikes of later origin than the diorites.

Along the border of the main diorite there are found outlying arms, usually running parallel with the structure of the enclosing rocks, but sometimes slightly cross-cutting. Occasionally, also, there are outlying stock-like intrusions which are not greatly lengthened in the direction of the general trend of the country. Some of these outside masses are connected with the main area by surface outcrops, while others are not visibly joined to the central mass.

Within an area of about 50 sq. miles adjacent to Juneau the diorites show several distinct though related types, and these range from rocks composed almost entirely of hornblende- to mica-diorite, quartz-diorite and granodiorite. In the Coast range near Skagway, true granites are found associated with the more common diorites.

The intrusive rocks in which the ores of the Treadwell mines occur is one of the extreme differentiation-phases of the diorite. It has been described by Dr. Becker as sodium-syenite, or albite-diorite.

The areal relations of the main diorite and the outlying masses in the vicinity of Juneau are shown in the accompanying sketch map, Fig. 3. Each separate intrusion appears in the field to have a distinct mineralogical composition.

Crystalline Schists.—Next to the main intrusive mass of the Coast range, but intricately dove-tailed by its offshoots and outlying arms, there is a series of crystalline schists derived by metamorphism from sedimentary rocks. These are mainly mica, hornblende and garnet-schists, such as would naturally result from the alteration of calcareous and feldspathic sandstones and shales. With them, however, there are some well-defined strata of limestone and quartzite in which the effects of metamorphism are ordinarily less apparent to the eye, though the former are often thoroughly crystalline. The schistosity which characterizes this series follows the surfaces of stratification, as may be seen in many places from the attitude of the secondary structure in reference to the persistent strata of varying composition, such as limestone and quartzite.

Locally, igneous material has been intruded into the schists

in the form of small aplite stringers, either cross-cutting the schists or following their plated structure, forming in some instances a rock of gneissic appearance, not readily distinguishable from gneissic phases of the intrusive diorite. Rocks of this sort, which may be called injection-gneisses, are confined to the vicinity of the diorite masses. In the main the schists are regarded, however, as having been formed by the crystallization of the originally sedimentary strata without important addition of outside material.

The crystalline schists form the country-rock for many large masses of diorite, and also enclose dikes of a dark colored igneous rock, which though greatly altered seems originally to have been gabbro. The intrusions usually follow the structure of the rocks rather closely. In places the invading dikes have been mashed and recrystallized, and the gabbros sometimes give rise to hornblende-schists which are not distinguishable from others of sedimentary origin, except in cases where gradation from the original into the secondary rocks can be observed. Some of the aplite-dikes which cut the schists are mineralized, and many independent stringers, gash-veins and lenses of quartz occur, and are often gold-bearing, but so far as observed the prospect of finding productive deposits in the schist-band is not particularly encouraging.

The width of the zone of crystalline schists varies from about 3 miles in the transverse-section through the Douglas Island mines, to zero in the vicinity of Berners bay to the north, where it is cut out by the gradual edging-over of the Coast Range diorite. Toward the southwest, for a distance of several miles, its width is somewhat greater than 3 miles, but it becomes narrower again farther down the coast toward Windham bay. The variation in width of the schist-band indicates the amount of cross-cutting by the diorite. Excluding the bodies of intrusive rock which it contains this series has an apparent thickness of about 15,000 ft., no trustworthy evidence of duplication of beds having been detected, though carefully sought.

Black Slates and Greenstones.—The outermost of the three principal lithological groups of the region is composed of alternating beds of greenstone and black slate, with occasional lenticular masses of limestone. Up and down the coast, as far

as observation has extended, these rocks occupy all of the mainland strip between the band of crystalline schists and the shores of Stephens passage and Lynn canal. They occur also on the adjacent islands of the Alexander archipelago, and though the western limit of the band has not been determined, similar rocks cover all the inland side of Admiralty island which, lying opposite Juneau and Douglas island, parallels the mainland for a distance of 70 miles. The whole band is thus not less than 15 miles wide, but the extreme width of the mainland portion is about 8 miles. From the boundary with the schist on the northeast to the far side of Douglas island the distance is about 7 miles.

In the vicinity of Juneau, four sub-zones, based upon the distribution of the greenstone, may be distinguished. On the inland side there is a band of black slate free from greenstones, a mile or so in width. Next to the slates, and dipping beneath them, comes a band 1.5 miles wide, composed mostly of greenstone schist, derived from the metamorphism of ancient surface-lavas, though with these there are several thin intercalated beds of black limy slate. The third sub-zone, beginning somewhere beneath the waters of Gastineau channel, extends to the base of the mountains back of the relatively low platform on the inland side of Douglas island, and its width is therefore about 1.5 miles. The landward two-thirds is composed of alternating beds of greenstone and slate, with the latter in excess, as may be observed along the upper end of the channel where the strata strike out into Douglas island; while the outer third is composed entirely of black limy slates. These last-mentioned black slates form the country-rock for a system of syenite-dikes, certain of which constitute the Treadwell ore-bodies. The rocks of the fourth sub-zone form the mountains of Douglas island. They are basaltic greenstones and greenstone breccias, evidently representing a great series of volcanic flows and agglomerates.

All the different parts of the slate-greenstone series lie in parallel position, striking northwest and southeast, and dipping toward the northeast in conformity to the prevailing structure of the region. The rocks are considerably, but not uniformly, metamorphosed. The slates, which now contain large amounts of graphite, were originally carbonaceous shales; the limestone

strata are often only partially recrystallized and sometimes give a strong odor of petroleum; the greenstone are locally crushed and changed to chloritic schists. Slaty cleavage and schistosity are normally parallel with the bedding of the rocks, though locally, where plication exists, there may be wide divergence of primary and secondary structures.

The age of the slates is known to be Paleozoic, from the presence of probably Carboniferous fossils in associated limestones at Taku harbor, about 20 miles southeast of Juneau. They therefore correspond in a general way with the Calaveras formation of California, which they closely resemble in lithology, metamorphism and structure. The Calaveras formation likewise forms the country-rock for gold-quartz veins later than dioritic intrusions, resembling those of southeastern Alaska.

The greenstones are mostly volcanic rocks which flowed out upon the surface at different times during the deposition of the sedimentary strata with which they occur. In their present condition they are similar to the rocks called amphibolites in the U. S. Geological Survey reports on the Mother Lode in California, which have also been considered to be volcanic rocks crystallized under surface conditions.

In the Juneau region the way in which the thick masses of these rocks are built up in layers, sometimes interleaved with thin sedimentary strata, the occurrence of vesicular beds, and of breccias made up of volcanic bombs and fragmental igneous material, all indicate their origin as surface volcanics. There are, however, some green rocks of very similar appearance, which seem to be intrusive, and the two sorts are usually indistinguishable, unless, as rarely happens, the fact of invasion and later origin can be established by evident cross-cutting of the stratification.

In the band of slates and greenstones, there are occasional masses of diorite related to the main intrusive rock of the neighboring Coast range. In general these are distributed irregularly in all parts of the series, though in the immediate vicinity of Douglas island the only large masses are on the northern end of Glass peninsula, which forms the landward side of Admiralty island opposite the lower end of Douglas island; and on Grand island in Stephens passage near by. A small intrusion occurs back of Sheep creek on the mainland

about 2 miles from Gastineau channel, but the only other occurrences known are in the series of dikes which form an important feature in the local geology of the part of Douglas island where the Treadwell mines are situated.

Other intrusive rocks are narrow dikes of basalt or minette usually cross-cutting the country, and a series of basic dikes and irregular masses noted mainly in the upper or inland black slate sub-zone of this band, though they are also found in the crystalline schists, as already stated, and they have been recognized in the band of greenstones on the northeast side of Gastineau channel. These dikes which follow the structure of the slates closely, but not absolutely, are usually highly metamorphosed, but they seem originally to have had the composition of gabbro.

Ore-Deposits of the Region.—The accessible portion of the mainland from Windham bay northwestward to Berners bay, including also Douglas island, is here designated as the Juneau gold-belt.

Of the three bands or groups of rocks which occur in this belt, the slate-greenstone band is pre-eminent in the number of quartz-veins and other forms of metallic mineralization which it contains. All the proved placer-deposits and all the mines and noteworthy prospects of the district are situated in the strip covered by these rocks. The comparative accessibility of the slate-greenstone band may account for the distribution of the active operations in part; but farther inland, prospecting in the crystalline schists and in the diorite intrusives has never given equal promise of valuable deposits.

The most prominent economic feature of the slate-greenstone band is a strong lode-system or complex of veins, traceable throughout the length of the belt. South of Berners bay all the best placer-ground, and all the lode-mines which have produced important amounts of bullion, excepting the Treadwell group and its former placers, are on this main system of veins. It compares in a general way with the Mother Lode of California, but follows more closely the structure of the rocks. Its position is in the upper black slate sub-zone of the slate-greenstone band, just above its contact with the uppermost greenstone beds of the next lower sub-zone (Fig. 4). The characteristics of the lodes may be studied in the Gold Creek and

Sheep Creek mines, which have been producing for many years, but are only now being developed upon the large scale which the extent and importance of the deposits demand.

In the vicinity of Juneau the second sub-zone, composed mainly of greenstone-schists, shows considerable mineralization in the aggregate, and there is at least one vein which may be traced for several miles with practical continuity, but no mines have been developed. The corresponding rocks both to the north and south give more promise.

Observations on the occurrence and distribution of mineralization in the third sub-zone, composed principally of black slates, with some interbedded greenstones, have been confined to Douglas island, where mineralized dikes of albite-diorite intrusive in the black slate, form the Treadwell deposits. Outside of these properties there are some stringer-leads of quartz and a certain amount of general impregnation in basic greenstones which are probably ancient lava-flows, but neither of these types of mineralization is likely to yield workable deposits within this band of rocks.

The greenstones which form the mass of the fourth sub-zone are in part highly mineralized. Southeast of the Treadwell mines, in Nevada creek which is the southernmost of the longer streams on the mainland side of Douglas island, these rocks have been altered to propylite by solutions which have permeated them; and a large mass of the rock has been impregnated with disseminated pyrite carrying small amounts of gold. This mineralized material extends for nearly 1.5 miles parallel with the strike of the rocks and not less than 1 mile across their trend. With the pyrite, lead, zinc and copper sulphides sometimes occur, and when these are present, assays show silver and an increased proportion of gold. The richer material is, however, largely concentrated in narrow zones marked by ill-defined fissures, which are ordinarily transverse to the general northwest trend of the mineralized mass. Only a small amount of quartz is present in the form of vein-filling. This area of mineralization has not been thoroughly prospected, and workable ore-bodies may eventually be found in it.

Another similar mass of pyrite-impregnated rock occurs in the same band of greenstones opposite the Treadwell mines. Here the altered and mineralized material outcrops in a zone

perhaps not over 300 ft. wide, and is traceable for at least a mile from southeast to northwest. Such prospecting as has been done has not given particularly encouraging results.

In all the outer portion of Douglas island the amount of quartz in veins is small in comparison with that occurring in other parts of the slate-greenstone band, and while other instances of pyrite-deposition than those mentioned are known, from present developments none of them seem to be of much promise.

Nature and Date of Veins.—The instances of veining, and the sulphide deposits which have been cursorily noted, indicate that the region in which the Treadwell mines are situated is one of the very generally distributed mineralization. Most of the occurrences of metallic sulphides which carry gold or silver, or both, are in veins largely composed of quartz, or in limited impregnations of the country-rock adjacent to such fillings. All the deposits, thus far productive, throughout the whole belt are of this nature, and the more detailed descriptions which follow will show the Treadwell ores to be closely related to the same type. The gold-quartz veins are regarded as essentially contemporaneous in origin throughout, because the pre-existing fractures are nearly all referable to two or three simple, but extensive, systems of fractures, and detailed examinations have not furnished evidence of more than one period of vein-deposition, such as reopening of fissures or faulting of veins by distinctly later fillings.

The only other type of metallic impregnation which has been recognized is exemplified in the altered and mineralized masses of greenstone on the outer side of Douglas island. Here the mineralization is regarded as mainly due to hydrothermal action, involving addition of materials, rather than to dynamic metamorphism bringing about a concentration of materials already present in the rocks.[9]

The occurrence of segregated veinlets in the otherwise thoroughly impregnated rock, requires either a period of migration and reconcentration following that of first deposition, or a second distinct period of mineralization. Known data do not

[9] Gold-Quartz Veins of Nevada City and Grass Valley District, California, by Waldemar Lindgren. *17th Annual Report U. S. Geological Survey*, Part II., p. 94.

permit a close correlation of either of these features with the widely distributed metalliferous quartz-veins of the region, though on the face of things, the original mineralization is more likely to correspond than the secondary veining.

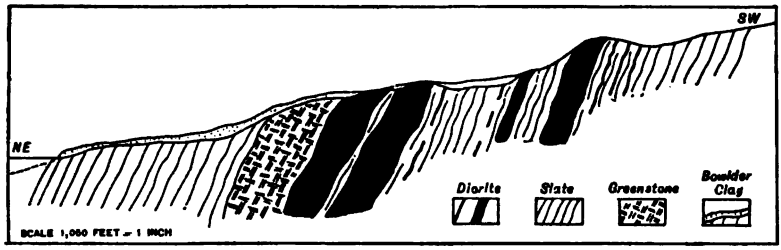
Without presenting the necessarily extended chain of circumstantial evidence involved in our present knowledge of the geological date of the gold-veins in southeastern Alaska, it may be stated that they are later than the great diorite intrusions now regarded as younger than middle Jurassic. They therefore correspond, in a general way, with the similar gold- and silver-veins occurring throughout the Sierra Nevada mountains of California and the geologically similar regions in Oregon, Washington and British Columbia. Their deposition is referred to a period of active water-circulation more or less directly consequent upon the invasion of the dioritic rocks occurring in the Coast range and outlying intrusions.

III. Geology of Treadwell Deposit.

General Features.—The Treadwell ore-bodies consist mainly of mineralized albite-diorite, occurring in the form of intrusive dikes in black slates, the structure of which they closely follow. These slates are metamorphosed shales in which both original bedding and slaty structure strike northwest and southeast, and dip about 50° on the average toward the northeast (Fig. 5). The ore-bearing dikes belong to a series of intrusions which appear interruptedly along the strike for a distance of about 3 miles, in a zone approximately 3,000 ft. wide. In the greater part of the intruded area, exposures are few, and only small dikes outcrop on the side toward the center of the island. On this side the zone seems to be irregularly limited, but next to the shore of Gastineau channel the border is defined by a heavy bed of greenstone running parallel with the slates and the intrusive dikes, and dipping with them toward the adjacent channel. The mineralized dikes which constitute the known minable ore occur just beneath this greenstone, which thus constitutes the hanging-wall both of the intrusion-zone and of the ore-bodies. Many of the dikes of albite-diorite, away from the hanging-wall have been greatly altered and impregnated with pyrite, but workable ore-bodies have not yet been discovered in them.

In the main, the strike of the different rocks is regular and being slightly oblique to the channel, the outcrops of the ore-

FIG. 5.

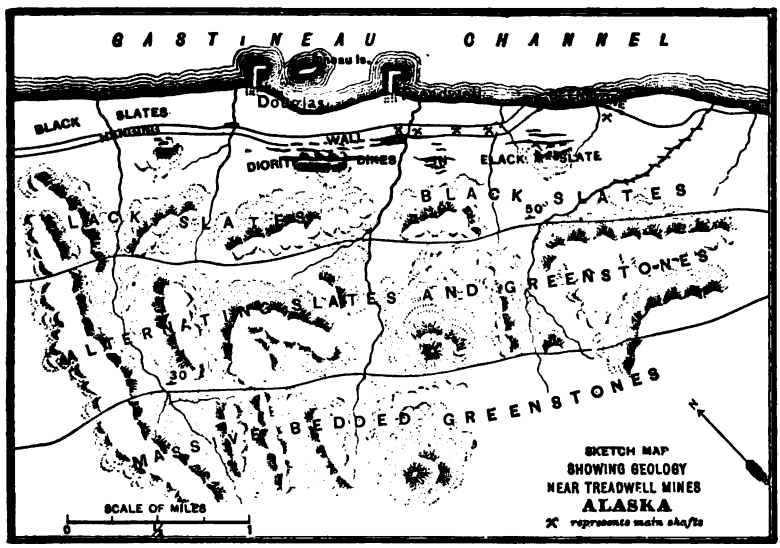


CROSS-SECTION THROUGH ALASKA-TREADWELL MINE AND NORTHERN SIDE OF DOUGLAS ISLAND.

Shows relation of ore-bodies and other dikes of albite-diorite to slate and greenstone.

bodies recede from the shore toward the northwest (Fig. 6). The base of the greenstone hanging-wall strikes the shore of the island about a mile below the Ready Bullion mine, at first running inland, and then back to a point below high-water

FIG. 6.



just beyond where the southernmost body of diorite is exposed in the open pits of the Ready Bullion mine. Reappearing within a few hundred feet it bends sharply and is next exposed

in the southeast pits of the Mexican mine. From this point it is traceable in a nearly straight line through the Seven Hundred Foot and Treadwell workings, and for a distance of several miles beyond.

In the vicinity of the mines there are no dikes of diorite on the channel-side of the greenstone, but about a mile to the northwest two croppings have been noted; and Juneau island, in Gastineau channel about 2,000 ft. from the foot-wall, is composed of similar rock which is somewhat impregnated with pyrite.

Besides the mineralization of the igneous dikes, the black slates of the same general belt on both sides of the greenstone-band contain occasional veins and systems of quartz stringers following the structure. Veining of this sort has been particularly noted along the foot-wall of the Treadwell greenstone for a distance of several miles beyond the mines. Assays of about $6 per ton in value have been obtained in some places, but there has been no systematic attempt to develop these stringer-leads, and their value is doubtful.

The rocks occurring in and near the mines, which will now be described in greater detail, are the following: The greenstone hanging-wall; the slate country-rock, enclosing both greenstone and ore-bodies; the dikes and lenticular masses of diorite, some of which constitute the ore; and a few small dikes of basalt.

The Greenstone.—The hanging-wall greenstone forms a prominent bed or stratum about 300 ft. in thickness where measured in the mines, but varying somewhat from this figure in different parts of its outcrop. So far as can be determined, it follows the structure of the slates, striking with them from southeast to northwest, and dipping northeast toward the near-by channel, beneath which it has been followed to a depth of about 900 ft. in the lowest workings. The outcrop is practically continuous for 4 miles northward from where the greenstone first appears on the shore of Douglas island. Then the bed thins out and is wanting for a few hundred feet, but it soon reappears and may be followed for an additional 2 miles, until it is lost beneath a heavy covering of vegetation.

As a rule, the rock is greatly altered, and in places it is even schistose or slaty, but portions are sufficiently unchanged to

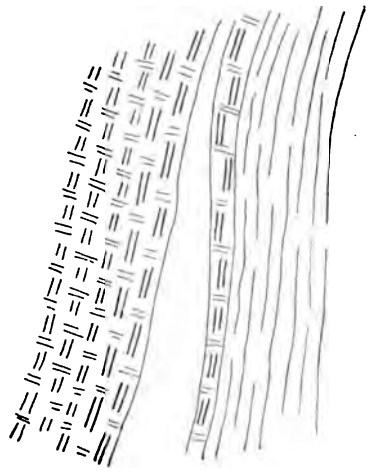
indicate the original composition and structure. In the vicinity of the Ready Bullion mine the rock is granular, consisting mainly of coarsely crystallized hornblende, though it contains a great deal of magnetite and some pyrite. A specimen from the Mexican workings, which might be called andesite, contains porphyritic crystals of plagioclase and augite in a decomposed ground-mass, which seems to have consisted largely of small prismatic feldspar crystals. The secondary minerals are chlorite, epidote, serpentine, and calcite. Beyond the workings toward the northwest, the greenstone is a fine-grained diabase.

The greenstone was called gabbro by Becker, who regarded it as later than the rock of the ore-bodies, but there is now sufficient evidence to establish the opposite age-relation, and reasons exist for doubting its intrusive nature. The inclusions of light-colored rock fragments in the greenstone, which form the basis of Becker's conclusions, are represented in his collection by a specimen and a thin section, showing a distinctly outlined fragment of grayish granitoid rock enclosed in greenstone; but the diagnostic value of this occurrence is open to doubt, since at several points in the region, pebbles and fragments of similar material occur in the volcanic greenstone breccias at different horizons in the series of interbedded slates and greenstones, showing the existence of an available source of granitoid material prior to the deposition of the slates and the outpouring of the contemporaneous lavas.

In the open pits of the Seven Hundred Foot and Mexican mines the exposed lower part of the greenstone-bed is very schistose, and this slaty rock forms both walls of the ore-body. Between the ore and the black slate usually forming the foot-wall, there is a plate or layer of chloritic schist of somewhat variable thickness, evidently identical with the schistose or slaty greenstone of the immediate hanging-wall, and the latter grades off into the massive rock (Fig. 7). This relation suggests that the locally developed schistosity of the greenstone existed before the intrusion of the diorite dikes, or was produced at the time of their invasion, and in either case the greenstone must be the older rock. More definite evidence in the same direction was noted in an old stope above the 220-ft. level in the Treadwell mine. Here the main mass of diorite

lies below all of the greenstone, but the latter is somewhat schistose, and a narrow offshoot from the diorite cuts across

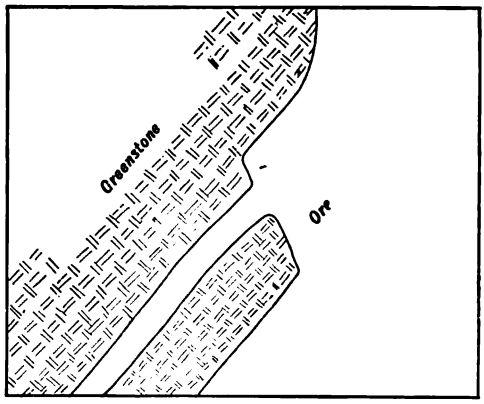
FIG. 7.



The diorite, which is left blank in the sketch, lies between walls of schistose greenstone, and a few feet distant from the slate which usually forms the foot-wall.

SECTION OF NARROW ORE-DIKE OF ALBITE-DIORITE OBSERVED IN SURFACE-PIT OF "700-FOOT" MINE.

FIG. 8.



Scale, 1 in. = 8 ft.

MINERALIZED DIORITE INTRUSIVE IN GREENSTONE OF HANGING-WALL AT TREADWELL MINE.

this secondary structure for a distance of about 3 ft., and then follows the schistosity parallel with the wall of the large ore-body (Fig. 8).

Without the above proof that the diorite is intrusive in the greenstone, several general considerations would lead to the probability of this relation. In the region at large the dioritic rocks invariably cut the bedded greenstones, and in Sheep creek they are even later than the gabbro dikes which follow the structure of the enclosing rocks approximately. None of the basic intrusives which are evidently later than the Coast range diorites show any tendency to follow the structural trend of the region, but, like the small basalt-dikes in the Treadwell mine, they characteristically hold to transverse courses. The way in which the greenstone limits the zone of diorite dikes, and the marked coherence of individual dikes to its lower surface, both point to the hanging-wall stratum as a controlling feature in the distribution of the diorite, and therefore suggest its earlier existence. The probability of this connection is well brought out by the detailed cross-section and map (Figs. 5 and 6). Again, if the attitudes of the diorite dikes and the greenstone in reference to the slate country-rock are compared, it is found that the diorite shows all the ordinary structural characteristics of intrusions, while the greenstone exhibits no features which necessarily require an intrusive origin. The diorite bodies change in shape from place to place, branch irregularly, cross-cut the stratification locally, and include masses of slate. The greenstone is a single layer or bed which continues along the same horizon for at least 6 miles, showing but slight variations in thickness; it does not cross-cut the slates so far as observed, and it contains no slate inclusions. Under the circumstances it is strongly believed that the greenstone is not intrusive, but that it originated as a lava-flow similar to many others in the same general series of alternating sediments and igneous rocks, while the diorite seems to have been intruded at a much later date.

The Black Slate.—The black slates, which constitute the main country surounding the Treadwell mines, belong to the third sub-zone of the slate-greenstone band already described. Together with the hanging-wall greenstone they constitute all of this sub-zone which appears on the southern half of Douglas island, the remaining portion being beneath Gastineau channel. They are highly metamorphosed, carbonaceous and calcareous shales, of fairly uniform texture; their stratification is usually

determinable from variations in color and from slight changes in the character of material, and so far as observed the bedding and principal slaty cleavage are always in accord.

The cleavage of the slates is regarded as having been produced before the syenite-intrusions, the direction of which it largely controls. In this respect the secondary structure corresponds with that of the sedimentary rocks of the general region, all of which were tilted and metamorphosed before the diorites of the Coast range were intruded. The slates do not appear to have been altered by contact-metamorphism next to the intrusive dikes of syenite.

The Albite-Diorite.—Classification of the Treadwell rock is somewhat difficult, because it has been impossible to secure entirely unaltered material. Dr. Becker, who first studied it with care, gave it the designation "sodium syenite," to distinguish it from the ordinary syenites which contain potassium as their alkali constituent. However, since the soda-feldspar albite, which is the characteristic mineral of the rock, belongs to the plagioclase series, and these feldspars are the distinguishing feature of dioritic rocks, he suggested the alternative name "albite-diorite," which is here employed because it serves to indicate the known relationship of the Treadwell rock with the dioritic intrusives of the adjacent Coast range.

The rock varies in mineralogical composition from place to place, but it is always very much changed from its original condition. Most of it shows little or no ferro-magnesian minerals, either because they were never present, or because they have been decomposed and carried away by the mineralizing solutions which have permeated the rock. Specimens were collected, however, which contained hornblende in apparently original prisms, and biotite is sometimes observed. Secondarily-crystallized mica and green hornblende are somewhat common, and with them a considerable amount of epidote is ordinarily found. Feldspar is present in two conditions, original and secondary. The primary feldspars of the magma were albite-oligoclase, occurring in phenocrysts now always clouded by decomposition-products, and microperthite with some pure albite, forming a granular ground-mass of distinctly later crystallization. The composition of the phenocrysts is inferred in general from the presence of epidote as one of the minerals

formed by the alteration of the feldspars, but this has been checked by the optical characteristics of relatively fresh material occurring in several specimens. The secondary feldspar is always albite, and is usually quite free from decomposition, and when it occurs in sufficient amounts it gives the rock a very fresh appearance. It seems to have been formed mainly at the expense of the original microperthite, which it replaces in part.

Quartz seems not to have been an original mineral in the albite-diorite, and it is never observed in the body of the rock associated with the secondary albite, but is confined to the veinlets which intersect the dikes. Calcite is common both in the veins and distributed through the rock itself along with the albite of the second generation.

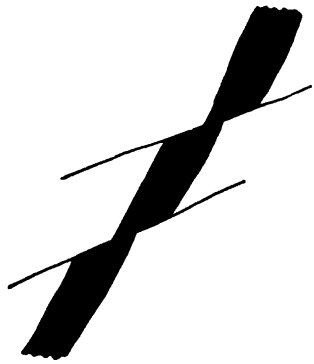
Original accessory minerals noted are: Apatite, titanite, rutile and magnetite. The secondary minerals which have been noted are: Uralite (secondary hornblende), green mica, chlorite, epidote, zoisite, calcite, quartz, sericite, rutile, pyrite, pyrrhotite, and stibnite, with other sulphides occurring exceptionally. Some of the magnetite seems also to have originated from the breaking up of former iron-bearing minerals, and where it surrounds cubes of pyrite it has apparently been deposited from the mineral solutions.

In the vicinity of the mines albite-diorite occurs in the black slates as dikes distributed throughout a zone about 3,000 ft. in width, and extending along the strike for a distance of 3 miles. Only bodies near the hanging-wall of this zone have been mined, up to the present time, though several others are strongly mineralized. The dimensions of the different dikes are extremely variable, the larger ones having a maximum observed width of over 200 ft. in surface-exposure and in the mine-workings. From this, all sizes occur down to the width of one's hand, and toward the ends of the intrusive area only small dikes occur, as may be observed along the bed of Bullion creek. The sketch map (Fig. 6) indicates the general distribution of observed dikes. Undoubtedly a still larger number, principally of small dikes, are hidden by gravel beds and by the deep mat of decaying vegetation which covers much of the ground. In many cases, and this is particularly to be noted in the dikes which have been mined, the individual in-

trusions are made up of a series of lenses formed by alternate bulging and pinching of the intrusive mass. In places the structure of the slate follows these irregularities, while elsewhere there is local cross-cutting, or even faulting. Pinching and swelling of the diorite is shown in both vertical and horizontal cross-sections of the dikes, though in general it is to be noted that the variations are more frequent and the changes take place within shorter distances upon the dip than upon the strike. These features are illustrated in Figs. 10 and 11, which, with the addition of a few details, have been selected from the working-maps and stope-sections of the different mines.

The greater frequency of the variations on the dip, which

FIG. 9.



IDEAL SKETCH SHOWING MANNER IN WHICH FAULTS OF LOW DIP MAY DISPLACE AN INCLINED DIKE TO GIVE APPEARANCE OF ALTERNATE SWELLING AND PINCHING.

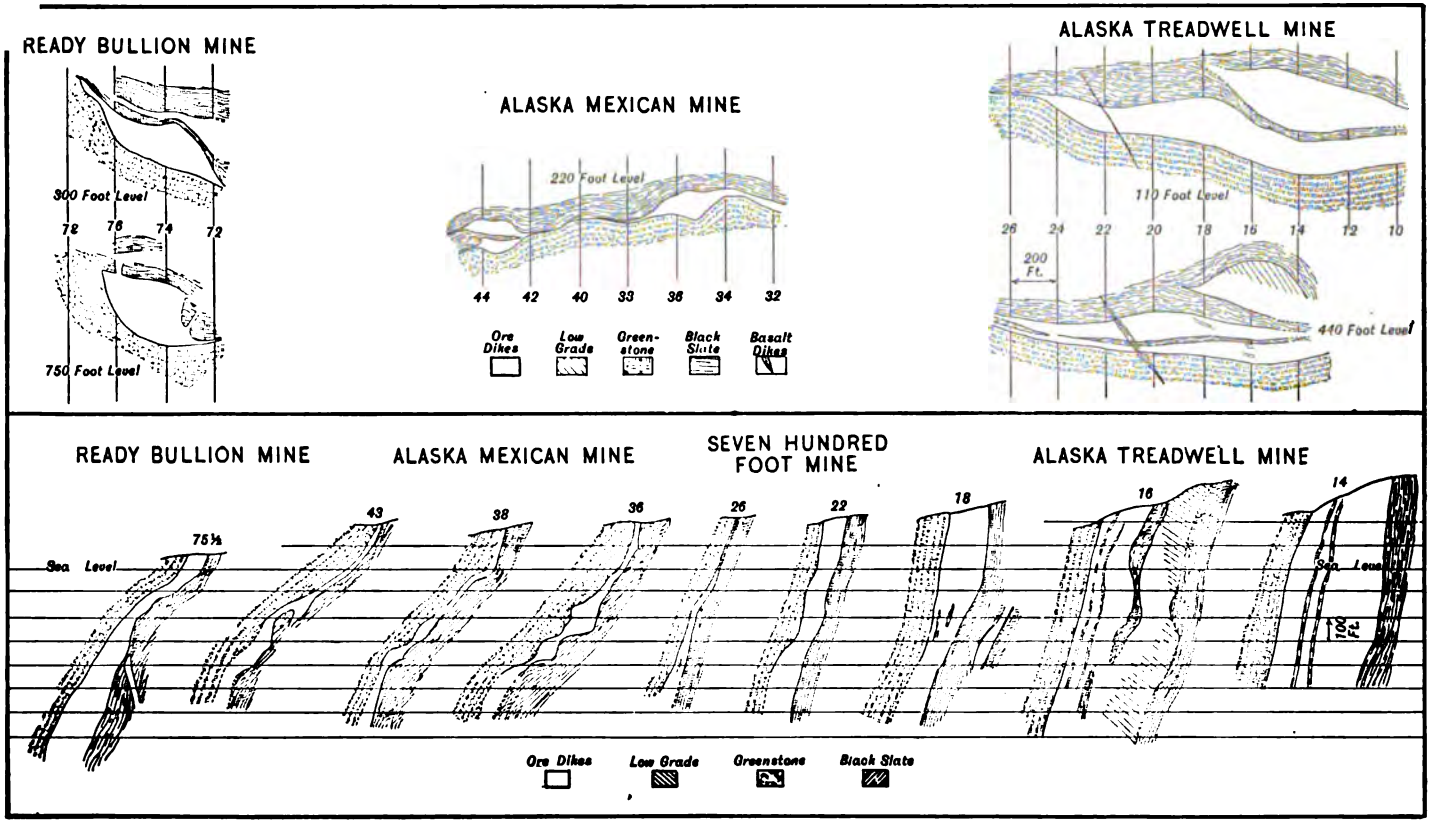
has been mentioned, may be due to faulting, for in the west end of the "Glory Hole" at the Treadwell mine, and in one or two other cases underground, where observations have been less readily made, the ore-bodies are offset by movement along surfaces which strike nearly parallel to the veins, but dip at a lower angle. A series of such faults would produce the effect of alternate swelling and pinching (Fig. 9).

Outside of the ground which has been worked, the details of the various diorite masses are unknown, but their general distribution is shown upon the geological map, and the generalized cross-section through the workings of the Treadwell mine indicates the relative number and size of the dikes which out-

crop (Figs. 5 and 6). Considerable work was done several years ago in prospecting adjacent bodies of diorite, many of which are as thoroughly impregnated with pyrite as the developed ore-bodies. So far as known the gold values are mostly very low, and while mines may yet be discovered, explorations have not thus far resulted in important discoveries.

The occurrence of the sulphide-bearing diorite which forms the Treadwell ore-deposit has been described by Dr. G. M. Dawson, who visited the mine in 1889. This geologist states his impression that the deposit represents the upper portion or "feather edge" of a granitic intrusion, probably contemporaneous and connected with the granites of the neighboring Coast range. The structural relations presented by this view are entirely in accord with present observations; for, while the rock cannot be strictly classed as granite, neither can a large part of the rocks which form the core of the Coast range be so classed, since their composition is usually dioritic. The diorite of the Douglas island mines doubtless belongs to the Coast Range period of intrusion, and if the small dikes of basalt which are found from place to place throughout the region be excepted, it is the youngest of the bed-rock formations in the vicinity. At the time of its intrusion the rocks which now appear at the surface occupied a position deep within the shell of the earth (lithosphere), and while many masses of the Coast Range diorite were forced through to the surface, it is doubtful whether any of these particular dikes ever extended as far as the surface which then existed. Taken together they represent intrusive material which was arrested en route, while larger masses of related rocks in the region are regarded as the once deep-seated portions of intrusions which probably had actual surface-exit. In the underground workings the blind endings of certain of the dikes show that some of them do not extend even to the present surface. How much farther the larger ones may have penetrated the slates now removed by erosion cannot be estimated.

Basalt Dikes.—In several places in the mine-workings basalt dikes, which cut all the other rocks, have been encountered. They are narrow, usually from a few inches up to 3 ft. in width, and have sharply-defined walls. Locally, the dikes occur in pairs, and in several places are seen to divide, par-



FIGS. 10 and 11.—PLAN-SECTION AND TRANSVERSE SECTION OF DOUGLAS ISLAND MINES.

ticularly when they occur in zones of sheeted rock. The fissures in which they occur are transverse to the strike of the rocks and trend from N. 10° W. to about north and south true meridian, with a rather steep dip toward the west. As a rule they are not mineralized to any important extent, though a small amount of pyrite sometimes appears, and occasionally they contain a considerable amount of this mineral. In several places veinlets of calcite occur along the selvage, but these are readily determinable as of later origin than the greater part of the quartz and calcite which form a reticulation throughout the mass of the ore-material.

IV. The Ores.

General Description.—The occurrence of the albite-diorite dikes which constitute the Treadwell ore-bodies has already been given. The ore consists mainly of rock impregnated with sulphides, principally pyrite, and in part shattered and filled by reticulated veins of calcite and quartz, which also carry sulphides. The ore-bearing dikes are considerably mineralized throughout, and often the whole mass can be mined. Locally, however, the values are too low to pay for extraction, and portions of the rock must be left.

Three sorts of ore are recognized by the miners, "quartz," "brown ore" and "mixed ore." The so-called quartz-ore, which constitutes the bulk of the workable material, is essentially mineralized diorite, but it usually contains calcite and quartz, the former disseminated or in veins, the latter only in veins. As a rule, its color is white or light gray, but in many places it has a greenish cast. The brown ore is derived from a comparatively small amount of productive mineralization occurring in the walls or in the narrow horses of slate, where the presence of gold-bearing sulphides is commonly recognized by a brown color, which leads to the popular designation of this ore. The brown material grades into the ordinary black slate, and its color is apparently due to decarbonization of the carbonaceous rock by percolating sulphide solutions. Impregnation of the slate is by no means always present, and where it occurs, it seldom extends for more than 2 or 3 ft. from the walls of the main ore-mass. The mixed ore, which is more abundant than the brown, is composed of slate intricately in-

truded by small dikes of very fine-grained diorite, the whole being impregnated with sulphides in the same way as the ordinary ore.

The value of the material mined varies from, say, $1 to $5 and even $10 or more per ton, though in the course of development a great deal of less valuable rock is extracted, and in working the open pits large amounts of worthless slate must be moved, much of which goes with the ore to the stamps. In general, the average value of the rock has been a few cents over $2 for the past two or three years. From 60 to 75 per cent. of the gold is free-milling, and the concentrates, which the mill-records show to be about 2 per cent. of the material treated, assay from $30 to $50 per ton.

Shape of the Ore-Bodies.—The impregnation of the dikes in which the ore occurs is, for the most part, so general, and the presence of at least small amounts of gold is so constant, that it is impossible to recognize any well-defined masses which may properly be distinguished as ore-shoots. Though the values are by no means uniformly distributed, from the assay-plan they do not appear to occur in any regular way; and indeed the distinction between ore and rock too lean to pay for extraction is often a matter of only a few cents. The actual differences in gold-tenor of several contiguous samples taken from the ore are usually much greater than the difference between the average of any considerable block of ore and the contents of intervening masses of poor rock. In several places mere joints or seams may be noted separating the ore and the poor material, and it frequently happens that blocks of the latter, which show assays from a trace up to $1, are entirely surrounded by ore averaging $2 or more. Structural limitations, such as joints, however, are difficult of observation, because the sides of the drifts are everywhere covered with dust.

The plans and cross-sections given in Figs. 10 and 11 represent the shape of the ore-dikes and their variations in form from place to place. These diagrams are taken from the working-maps and stope-sections compiled by the engineers of the company. The sections are numbered to correspond with the lines 100 ft. apart, on the general map of the mines given in Fig. 1.

In general, the best ore is that which contains the greatest

number of quartz and calcite veinlets, and though their absence is not an infallible indication of valueless material, it seems that the irregular distribution of the gold has resulted mainly from original differences in the amount of crushing and the consequent varying permeability of the rock. Where the metasomatic replacement of the diorite by secondary albite is absent, the sulphides usually replace such minerals as hornblende or mica, and it is suspected that in these cases the gold-content is ordinarily low.

In planning the position of stopes, the assay-charts often enable locating the pillars in relatively poor material, but as a rule the low-grade rock is not found to persist for the whole distance between two mine-levels. The largest masses, which have been left because of their leanness, are on the foot-wall side of the south dike in the Treadwell workings, but even here there are great variations in the gold tenor at different places. On the 110-ft. level all the rock was minable; on the 220-ft. level from 10 to 40 ft. of low-grade stuff was left on the foot, excepting for a distance of about 150 ft. On the 330-ft. level good values were found up to the slate, excepting for 200 ft. along the west end, where 20 ft. or so were left; while on the 440-ft. level not over half of the rock gave assays above $1. The relative position of the high-grade and low-grade material in this part of the mine is shown in the plan of the 440-ft. level and in section 16 given in Figs. 10 and 11.

Persistence in Depth.—The ore-dikes have been developed along the dip for a distance of, approximately, 1,000 ft. in all three of the mines now operated. The Treadwell workings reach about 700 ft. below sea-level, the Mexican 600 ft., and the Ready Bullion 800 ft. In no case has it been possible to make out any progressive change in the character of the ore, as depth was attained. The assay-charts show the ore in the lowest levels to be quite as good as in the upper workings, and it is evident that variations along the dip are not greater than those observed from place to place along the strike. It is true that the mine-records for a period of years show a gradual decrease in the value per ton of the material which has been treated. This is especially noticeable in the case of the Treadwell mine, which has been the longest in operation, but it is the result of increasing the tonnage by mining low-grade rock, rather

than an indication that the average value is decreasing with depth.

It seems fair to assume that the ore will continue to a considerably greater depth without important change in average value. There is nothing in the character of the ore to indicate any important secondary concentration of values by oxidizing-waters near the surface. On the other hand, the characteristics of the deposits are believed to indicate that it was formed in its present condition by the direct action of ascending-waters. If this idea is correct, there can be little doubt that the mineralization and the values will continue to a much greater depth than has been reached, and it may be fairly anticipated that the limit of deep mining will finally depend more upon increased costs attendant upon hoisting and pumping than upon the exhaustion of the ore.

Veining in the Ore-Bodies.—In almost all parts of the Treadwell deposit reticulated veinlets of calcite and quartz are prominent features of the mineralized dikes. The veinlets are often composed entirely of calcite, but this mineral is usually accompanied by quartz, though the latter seldom, if ever, occurs by itself. The veinlets are rarely more than a few inches in width, many are only a fraction of an inch across, and the microscope shows the presence of minute fracturing between the veins visible to the naked eye. The veins are usually closely spaced, and an estimate based on a study of all the mine-workings indicates that infiltrated materials make up nearly one-fifth of the mass of the ore.

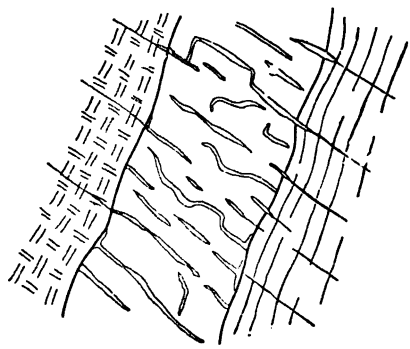
The boundaries of the veinlets against the enclosing rock are sometimes distinct, but in many cases there is an apparent gradation from the vein-matter into the altered diorite. When the amount of introduced minerals is large in proportion to the mass of the matrix, in small specimens it is often difficult to distinguish the vein-stuff from the rock; though in large fragments or on the stope-faces, the general extent of the different portions of the ore-material is exhibited. The microscope shows that the merging of the interstitial veinlets with the rock which they cut is due to penetration of the latter by calcite, which is intercrystallized with secondary albite, formed at the expense of the original feldspar.

Veinlets traverse the rock in different directions, but the

greater part of the filling occurs in fissure-like openings constituting two well-marked systems. One set of fractures strikes and dips approximately with the structure of the enclosing slates; the other, which is the more prominent, strikes slightly oblique to the structure of the country and dips in the opposite direction, that is, toward the southwest.

In places where the mineralized dikes are narrow, the set of fissures parallel to the country-structure usually diminishes in importance, and often only the cross-fractures have been developed. This may be explained upon the supposition that the tendency to motion parallel to the walls of the intrusions was taken up outside of the massive rock in the slates, while the

FIG. 12.



This dike, about 3 ft. wide, is cut by veinlets of calcite and quartz, which fill transverse fractures. The breaks extend into both walls, but the filling never extends far beyond the dike itself.

DIKE OF ALBITE-DIORITE IN OPEN CUT OF READY BULLION MINE.

transverse strain affected both the slate and the intrusive rock, the latter being specially susceptible to cross-fracture because of its small mass and brittle nature. Cross-fractures, filled with vein-stuff and limited to a narrow dike in the slates, may be seen to good advantage at the east end of the Ready Bullion pit, near the southernmost outcrop of the diorite (Fig. 12). Throughout the mines it is the rule that all transverse gash-veins stop at the walls of the diorite, and while there are a few exceptions the quartz seldom penetrates the country-rock to any great distance, and when it does it diminishes rapidly in thickness. However, this is not always due to the non-continuance of the fissures, for they may be frequently observed con-

tinuing from the diorite into the slate in the form of well-marked joints.

Gangue-Minerals.—Feldspar, calcite and quartz are the three important non-metallic minerals of the Treadwell ores. Part of the original feldspar of the intrusive diorite remains in the ore, and with a considerable amount of secondary feldspar forms the principal gangue-mineral. Other minerals of the unaltered rock are hornblende and mica, but these are present in relatively small amounts, as is also epidote, which has been formed as a product of alteration from them. Calcite and quartz occur in veinlets penetrating the diorite, and make up perhaps one-fifth of the material which is mined. Calcite is also found disseminated irregularly in the more altered parts of the diorite, unaccompanied by quartz. When sulphides and calcite are both present, they are almost invariably in contact with each other, but the secondary feldspar also carries a great deal of pyrite.

The occurrence of ferruginous calcite is common in the superficial workings, where it may have been formed by the action of iron-bearing solutions upon the primary calcite of the deposit. It occurs also in small amounts in deeper parts of the mines, where it is possibly an original mineral. A small amount of pink carbonate, probably a mixture of calcite and rhodochrosite, has been observed in the open pits of the Ready Bullion mine.

Metallic Minerals.—As shown by the mill-records, the metallic minerals, or "sulphides," constitute about 2 per cent. of the Treadwell ores. They consist mainly of pyrite, and a considerable amount of magnetite is also present.

Pyrite occurs both in the rock and in the veinlets, but the position of the sulphides has no apparent influence on the gold-content. In the rock it invariably has the form of minute cubes, from a size scarcely visible to the unaided eye up to about one millimeter, rarely larger. It is distributed sparsely through the diorite accompanying the secondary minerals, especially the albite and calcite, though where these are not present, it is associated with epidote and uralitic hornblende. In the reticulated veinlets the pyrite occurs either as separate cubes, often several millimeters across, or in bunchy aggregates, forming "turkey-egg rock" which usually contains a value higher than the average.

Magnetite occurs only in the form of minute grains outside the veinlets. Part of it appears to have been an original constituent of the diorite, but much of it was deposited secondarily along with the pyrite, perfect cubes of which it sometimes surrounds.

Stibnite occurs in minute needles, and though seldom visible to the naked eye the microscope shows that it is widely distributed in various parts of the mines. As a rule, it occurs imbedded in calcite, but it is sometimes in the secondary albite. Pyrrhotite often accompanies or takes the place of the pyrite and may be readily isolated from the concentrates by means of a magnet. Chalcopyrite, galena and sphalerite occur sporadically; and native arsenic, realgar and orpiment have been noted in small quantities. Assays are said to indicate the arsenical nature of much of the pyrite, though the presence of true arsenopyrite has not been recognized. Molybdenite is frequently noted, though it is irregularly distributed.

Occurrence of Gold.—Visible gold has been observed in veins of coarsely crystalline calcite enclosed in the ore-bodies. This occurrence is, however, rare; and in general even the microscope does not reveal the form in which the precious metal exists. I have not been able to distinguish gold in the thin sections studied under the microscope, but Professor F. D. Adams, who examined the material collected by Dawson in 1887, observed gold mechanically enclosed in crystals of pyrite. It is evident that a considerable amount of gold must be in the metallic condition, because a large proportion is saved by amalgamation, the amount sometimes being as high as 75 per cent. of the total assay-value.

The gold is perhaps mainly associated with pyrite, but rather coarse crushing is the present mill-practice,[10] and so much of the pyrite passes the screens in comparatively large grains or unbroken crystals, that it seems open to doubt whether from 60 to 75 per cent. of the gold could be free-milling if it were all associated with the iron sulphide. The non-amalgamating portion undoubtedly does occur with the pyrite, because the concentrates contain only pyrite and magnetite, with a small amount of pyrrhotite; all the stibnite and molybdenite going

[10] Slot-screens equivalent to 18- and 20-mesh wire screens are used.

into the tailings. The portion which amalgamates readily may occur with the stibnite or with the gangue. Molybdenite can hardly be an important carrier of gold, because it seems to be somewhat limited in distribution, although its presence in visible quantities is said to indicate high values.

As a rule, the values vary with the amount of interstitial vein-matter, but the position of the pyrite in the rock or in the quartz and calcite seems to have no influence upon the amount of gold. In some places, where the ore is of average grade, all the metallic minerals seem to be in the rock; and careful search is necessary for the discovery of any sulphide in the quartz or calcite. Elsewhere the sulphides may be almost entirely confined to the veinlets. A limited amount of material is mined which contains practically no stringers of quartz or calcite, the sulphide being disseminated through the mass of the rock, for instance, in the cross-cut on the 440-ft. level and in stope No. 1 of the 330-ft. level in the Treadwell mine. In other places material of similar appearance, containing an equal amount of pyrite, yields only a very small amount of gold.

Metasomatic Alteration.—As already stated, the Treadwell ore-bodies are dikes of albite-diorite filled with reticulated veinlets of quartz and calcite, and permeated with metallic sulphides carrying small amounts of gold.

From the structure of the deposits it is evident that the dikes were subjected to pressure which caused fracturing, whereby they became porous, affording channels of easy circulation for underground waters. The minerals in the ores and their mutual relations suggest that carbonated and mineral-bearing solutions found the broken dikes and continued to move through them for a very long period. In transit these waters attacked the minerals of the albite-diorite, decomposing them, and in some cases effecting more or less complete metasomatic replacement. As a rule, the hornblende and mica of the original rock have entirely disappeared, their place being taken by aggregates of secondary minerals, sometimes including metallic sulphides. A few specimens of relatively unaltered material indicate that the original rock characteristically contained two sorts of feldspar, albite-oligoclase and microperthite. The first occurs in phenocrysts of fairly definite form, often showing concentric structure, and always clouded

by decomposition-products, except the usually narrow rims, which are clear. The microperthite, which has the characteristic mottled appearance of this minute intercrystallization of albite and orthoclase, is entirely interstitial as regards the albite-oligoclase. It is usually nearly free from decomposition-inclusions, and is ordinarily accompanied by some clear albite. When pyrite occurs in such slightly altered material it lies in or next to decomposed hornblende crystals. Most of the rock has suffered considerably more alteration, and pyrite occurs throughout the interstitial ground-mass. Its introduction has apparently been accompanied by a breaking-down of the microperthite, for this mineral, so abundant in the comparatively fresh rock, is usually entirely absent when the sulphide occurs outside of the decomposed hornblende, that is to say, in the interstitial feldspar. In the most altered rock the place of the microperthite is taken by an aggregate of small albite crystals, and this mineral is regarded as a secondary replacement of the original feldspar. In some cases the replacement has gone so far that the crystals of albite-oligoclase have been attacked. This feature is relied on in part to prove the secondary nature of the albite, for intermediate stages, in which the microperthite is only partially replaced, were not noted during the preliminary study of the thin sections. Still more conclusive evidences that the albite is of secondary origin are its occurrence in veinlets cutting the old feldspar; the fact that it is found intercrystallized with calcite, both in veinlets and throughout the rock itself; and the fact that where albite forms the interstitial material instead of microperthite, pyrite and often stibnite are present, imbedded either in the feldspar or in the evidently contemporaneous calcite.

The alteration of the Treadwell diorite is regarded as a phenomenon which accompanied the formation of the veinlets intersecting the rock, and the metasomatic action is attributed to the same solutions as those which deposited the quartz and calcite. The minerals last named appear to have been for the most part introduced, but the albite is believed to have been formed entirely, or nearly so, from the previous minerals of the diorite, because it is not found in the larger vein-fillings. It is commonly observed that where both calcite and quartz are present in the fractures, the former usually occurs next to the walls, and it always permeates the rock to a greater or less ex-

tent. According to Lindgren, alteration of the sort here described has not been previously recorded, for though albite occurs as a vein mineral in California, it has not been detected among the metasomatic minerals in the wall-rocks of veins.[11]

In this connection, however, reference should be made to pseudomorphs of albite after adularia from St. Gotthardt. These are described by Bischoff,[12] who gives an extended discussion of the probable chemical reactions involved, and suggests the competence of waters containing sodium chloride to effect the observed replacement of potash feldspar by soda feldspar.

The occurrence of values in the wall-rock to such an extent as is observed in the Treadwell ores is also somewhat unusual, though not unique. So far as it was possible to ascertain, the position of the sulphides in the rock or in the veinlets has no influence upon the gold tenor.

The Rôle of the Basalt Dikes—In his discussion of the genesis of the Treadwell-Mexican ores, Dr. Becker leaves some doubt as to the importance which he desired to assign to the basalt dikes as mineralizers. He first says that the genesis of the ores is probably connected with the dikes, but afterwards suggests the relative unimportance of their influence.[13]

In the Treadwell and Seven Hundred Foot mines, two narrow dikes of the basalt are observed in a zone of sheeting, which is undoubtedly later than most of the veinlets in the ore-mass, Fig. 10. A small amount of calcite is found along their selvages, but they contain little or no pyrite. Upon the west or hanging-wall side, the ore is somewhat richer than it is between and beneath them, but it seems that this variation in gold tenor cannot be attributed to the dikes as mineralizers, because the rock between them is not enriched, as might be expected had they been an actual source of gold. It seems likely that a rearrangement of values by relatively recent circulation has been going on, and the course of the currents may well have been controlled by the zone of sheeting in which the dikes occur, but secondary migration of this sort must be distinguished from the original mineralization, the extensive results of which in the neighborhood are entirely beyond comparison with the effects di-

[11] W. Lindgren, Metasomatic Processes in Fissure-Veins. *Trans.*, xxx., 578.

[12] *Chemische Geologie*, vol. ii., pp. 409–411.

[13] *18th Annual Report U. S. Geological Survey*, 1898, Pt. III., p. 69.

rectly or indirectly attributable to a pair of narrow dikes of this sort. It is now believed that they have no connection with the formation of the ore.

Other basaltic dikes occurring in Gold creek are regarded as practically of the same age as those on Douglas island, and these are also unmistakably younger than the gold-bearing quartz veins of that neighborhood.

Origin of the Fractures.—Upon the fracturing of the Treadwell dikes their impregnation with gold-bearing sulphides is evidently dependent. The systematic arrangement of the reticulated veinlets in two main sets standing at right-angles to each other and dipping in opposite directions, led Becker to the conclusion that the fractures had been produced through compressive shearing-stresses. He suggested that these stresses were caused by nearly tangential forces acting in a direction normal to the common strike of the two sets of fractures, which is also approximately the strike of the country-rock.[14] The fact that the fractures are due to compressive thrust need not be questioned, since the theory of the subject has been so ably developed and so fully corroborated by experiment.[15] Some doubt arises, however, as to the direction in which the forces may have been applied, because the geological history of the general region since the diorite intrusions seems to indicate that no widespread lateral compression has taken place. If tangential shortening has been going on, evidences of the fact, independent of the fracturing, has not yet appeared. On the other hand, a study of the wide physiographic features of this portion of North America has shown that a succession of continental uplifts has taken place since the period of the diorite invasion, and it seems necessary to suppose that such radial movements would tend rather toward areal dilation than toward contraction, as in the opposite case of tangential compression.[16]

It is suggested that the general fissuring throughout the Juneau district may have been caused by gravitative adjust-

[14] Geo. F. Becker, *op. cit.*, p. 67.

[15] Finite Homogeneous Strain, Flow and Rupture of Rocks, by Geo. F. Becker. *Bulletin of the Geological Society of America*, vol. iv., 1893, p. 13. Daubree, *Études Synthétique de Géologie Expérimentale*, p. 316.

[16] The Pacific Mountain System in British Columbia and Alaska, by Arthur C. Spencer. *Bulletin of the Geological Society of America*, vol. xiv., pp. 117–132 (1903).

ment in the rock-masses, tending to restore internal equilibrium disturbed during the uplifts which are known to have taken place. The rocks of the district consist of alternating beds of greatly varying physical character, and they possess an eminent cleavage-structure parallel with the stratification. Under stress, such rocks would yield more readily along the pre-existing structure-planes than in other directions. That this old structure has, in fact, taken up most of the internal movement during the later deformation of the rocks, is evident from the occurrence of so large a majority of the veins in parallel position with it; and it may be supposed that this control has prevented the formation of a large number of fissures in various directions, which would have resulted in the case of homogeneous or massive rocks deformed under their own weight.

Having been subjected to the same pressures as those which fractured the other rocks of the region, it is only natural that the Treadwell dikes should be broken along lines parallel with the general fissuring, and one of the two sets of veinlets occurring in the ore-bodies practically coincides with the structure of the enclosing slates. The other set, which stands at right-angles to the first, is not nearly so well develeped in the country-slates, probably because these yielded by bending, since they are very flexible when compared with the brittle rock of the dikes.

Source of the Vein-Forming Waters.—The formation of the Treadwell ores is assigned to the same general cause as the other veins of the region. Both are attributed to circulating-waters moving through channels opened by a general fracturing of the rocks.

Ascending-waters may be assumed from the nature of the metasomatic changes which they have effected, and also from the large amounts of carbon dioxide which they evidently contained.[17] That the waters were hot may also be safely predicated, because the erosional history, as well as the present topography of the region, both indicate that the veins now exposed must have been deposited at great depths, certainly from 6,000 to 10,000 ft. below the former surface and possibly very much

[17] The Gold-Quartz Veins of Nevada City and Grass Valley Districts, California, by W. Lindgren. *17th Annual Report U. S. Geological Survey*, Pt. II., p. 173.

deeper. The occurrence of tourmaline in some of the veins of Gold creek, and the occasional presence of fluorite elsewhere, suggests a connection with igneous emanations; since these minerals are characteristic of pneumatolytic action as exhibited in the case of tin-deposits and in various instances of contact-metamorphism. The presence of these minerals cannot be pushed to the value of evidence, because neither of them has been universally observed in the district, but, even for those who hold the theory that the final source of mineralizing-water is mainly meteoric, their occurrence may be admitted as probably significant of at least accessory contributions to the vein-forming solutions from igneous sources.

It is concluded that known facts do not lead to a recognition of the actual source of the solutions which have formed the mineral deposits, and for any present idea of their origin recourse must be taken in the direction of speculation. I am inclined to believe that the very wide occurrence throughout southeastern Alaska of intrusions related to, and of practically identical date with, the Coast Range diorites strongly indicates the possibility of a great buried *couche*, or reservoir, of igneous rock underlying the whole region. It is evident throughout the field that the veins were formed at a period subsequent to the invasion of the diorite, and they were probably formed long after intrusion had ceased; but it is not a violent supposition to consider that the deep-seated magma, from which the masses now observed at the surface had been given off, remained in a molten condition for a very long time.

A plausible hypothesis for the formation of the veins, based upon the foregoing ideas, is that the unknown forces, which at various times have caused general elevation throughout the region, were transmitted to the other overlying rocks by this great residual magma. In adjusting themselves to the changed conditions of equilibrium the rocks were fractured; then, as the deep-seated magma gradually cooled and crystallized, water and gases expelled from it found their way into the overlying rocks, and, searching out the easiest routes of travel along existing fractures, escaped to the surface.

Undoubtedly, waters of this origin might carry in solution all the elements which have been observed in the veins, and they would deposit their mineral-contents under various condi-

tions, viz.: from decrease of dissolving power through diminishing pressure and temperature; from precipitation through metasomatic interchange with wall-rock materials; or from precipitation due to mingling with solutions of some other derivation.

V. SUMMARY.

1. The formations of the mainland of southeastern Alaska are thrown into three lithological groups, which are distributed in parallel zones following the general trend of the coast. Two of these groups, the schists and the slate-greenstone band, are mainly metamorphosed sediments; the greenstone-beds, however, representing ancient volcanic flows of andesite and basalt. The third is the great complex of intrusive granular rocks, mostly dioritic, forming the mass of the Coast range. The general structure of the region is monoclinal, strikes being usually northwest and southeast and dips always toward the northeast.

2. The region is one of very general mineralization which has taken place since the diorite intrusions, and there is a probable correspondence in date with the extensive gold-veins of the Pacific States.

3. The Treadwell deposits appear to have been formed during the general activity of vein-forming solutions throughout the region. The ore-bodies consist of mineralized dikes of dioritic rock lying between a greenstone hanging-wall and a slate foot-wall, with a few detached masses near by in the slate. Hot ascending currents of water are regarded as the source of the minerals which have been introduced into the dikes, where they fill fractures and also replace the original minerals of the rock. The origin assigned leads to the expectation that the character of mineralization will not change with further depth, and deep mining is more likely to be limited by increasing expense of hoisting ore than by gradual decrease in gold tenor.

4. Speculation as to the cause of fracturing and the source of vein-forming waters, for the region at large, leads to the suggestions that the former have been attendant upon recognized continental uplifts in the region since the invasion of the Coast Range diorites, and that the latter emanated from a great reservoir of igneous rock, underlying the whole region at great depth, during its consolidation from a molten condition.

The Genesis of the Copper-Deposits of Clifton-Morenci, Arizona.*

BY WALDEMAR LINDGREN, WASHINGTON, D. C.

(Lake Superior Meeting, September, 1904.)

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THE following pages are a résumé of some of the conclusions reached during a study of the copper-deposits near Clifton. The field-work was finished in 1902 and a complete report is now in the press. A preliminary description of the district was published in 1903, in *Bulletin U. S. Geological Survey, No.* 213, pp. 133–140.

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The Clifton mines, always important, are at present the largest in the territory, the output in 1902 having reached 50,000,000 lb. of copper, chiefly divided between the two principal producers, the Arizona and the Detroit copper companies. The Shannon Copper Co. also contributed to this figure, and its output has greatly increased since then. The total output of the district to the end of the year 1903 is estimated at a value of about $49,000,000.

Topography and Geology.

Clifton is situated on the San Francisco river, a few miles above its confluence with the Gila river, in the southeastern part of the territory, and not many miles from the New Mexican boundary. An irregular mountain region, without well-defined ranges, lies here north of the broad, detritus-filled valley of the Gila river, which has an elevation of about 3,000 feet. The highest elevations in the mountains scarcely attain 8,000 feet.

Between the San Francisco river and Eagle creek, both tributaries to the Gila river from the north, a core of older rocks of about 70 sq. miles is exposed, consisting of pre-Cambrian granites, Cambrian quartzites, Paleozoic limestones, and a capping-formation of Cretaceous beds—all intruded by post-Cretaceous granitic porphyries. This older core, which seems to represent the broken-down edge of the great plateau-province, is completely surrounded and largely covered by volcanic flows of Tertiary age, including basalts, andesites and rhyolites, which have been extensively eroded; hence the lack of regularity so plainly apparent in the mountain complex.

The copper-deposits are all contained in the older rocks and distinctly antedate the Tertiary lavas.

The sedimentary rocks rest on a basement of red, coarse granite, forming two great buttresses, the Coronado and the Copper King mountains, both rising more than 3,000 ft. above the San Francisco river.

The Paleozoic series consists of the following-named formations:

Modoc limestone. (Lower Carboniferous.)	Up to 200 ft. thick, heavy-bedded, gray limestones of great purity, with some equally pure dolomites at the base.
Morenci shale. (Devonian?)	100 ft. of clay-shales, sometimes underlain by 75 ft. of argillaceous limestone.
Longfellow limestone. (Ordovician.)	Up to 400 ft. of limestones, partly cherty and, near the base, containing sandy beds. Some of these strata are dolomitic.
Coronado quartzite. (Cambrian?)	200 ft. of reddish, quartzitic sandstones resting on granite. Lowest member is a quartzitic conglomerate.

The three upper divisions contain characteristic fossils, while only a few small *Lingula* shells have been found in the Coronado quartzite.

The Cretaceous series rests unconformably on the Modoc limestone and consists of a succession of clay-shales and dark sandstones at least 200 ft. thick. Scant fossils indicate that it belongs to the Fort Benton horizon.

Gentle dips, rarely above 20° and generally directed westward, characterize the sedimentary rocks.

All of the above-mentioned rocks are intruded by a great stock of porphyry which extends in a northeasterly direction between the foothills near Eagle creek across to the great Copper King granite ridge overlooking San Francisco river. The main stock, which is about 8 miles long and up to a mile and a half wide, breaks up, at the southwest end, into a network of irregular dikes and sheets, and, at the northeast, into a system of northeasterly-trending dikes cutting through the granite. Laccolithic masses of porphyry occur in the Cretaceous shales and sandstones. The rock of the main stock ranges from a granite-porphyry to a quartz-monzonite porphyry. The sills and laccoliths are usually composed of diorite-porphyry, but the different types of rock are clearly facies of the same magma, connected by transitions and forming a single geological unit. Dikes of diabase occur in a few places.

The intrusion of the poryhyry took place during the latest Cretaceous or the earlier Tertiary, and was accompanied by great disturbances in the immediately adjoining rocks, particularly well noticeable in the Paleozoic sediments; but these disturbances of the strata did not extend far from the contacts.

The intrusion of the porphyry was followed by important

structural movements. The surface of the granite, together with the whole covering sedimentary series, was buckled into dome-shaped folds and then extensively fractured, the blocks sinking successively deeper towards the valley and settling unequally around the two great buttresses, or "horsts," the Coronado and Copper King mountains, the maximum throw of the normal faults being 3,000 feet.

An epoch of erosion followed, but floods of Tertiary lavas soon surrounded and largely covered the old rocks which have only lately emerged by the energetic action of a second epoch of erosion. The effects of the large faulting-movements, which preceded the lavas, are still visible in the greater topographic features of the region, especially in the bold escarpment of the Copper King ridge.

Occurrence and General Features of the Ore-Deposits.

The geographical distribution of the copper-deposits is practically coextensive with the great porphyry stock and its dike-systems. The deposits occur either in the porphyry or close to its contacts, or along dikes of porphyry in some other rock. Areas in which no intrusions have taken place are practically barren. This intimate connection with the porphyry is certainly a most important fact. There is only one small division of deposits which deviates from this rule—namely, that connected with the diabase-dikes.

Practically all types of deposits contain copper as the most valuable metal. Gold and silver occur, as a rule, only in minute quantities, except in some of the outlying districts where they become of more importance. The two most important mining-centers, Morenci and Metcalf, which are 3 miles apart, are both situated at the main contact of the porphyry stock and the series of Paleozoic limestones. Elsewhere the intrusive rock generally adjoins granite or Cretaceous sediments.

The ores consist of chalcocite, chalcopyrite, malachite, azurite, chrysocolla, brochantite, cuprite, and native copper. Covellite and bornite are practically absent. Brochantite, a basic copper sulphate, is very commonly present, especially in the oxidized veins in porphyry; and, in fact, constitutes in places an important ore. On account of its similarity to, and intimate intergrowth with, malachite it has usually been overlooked.

The following-named minerals have been found:—Native copper, native gold, quartz, chalcedony, rutile, magnetite, hematite, limonite, pyrolusite, coronadite (a new mineral, chiefly PbO and MnO_2), cuprite, pyrite, chalcopyrite, zinc-blende, galena, molybdenite, chalcocite, diopside, tremolite, garnet, epidote, muscovite, chlorite, serpentine, asbestos, kaolin, willemite, calamine, dioptase, chrysocolla, copper pitch ore, morencite (a new mineral, chiefly a ferric silicate), calcite, dolomite, zinc carbonate, malachite, azurite, libethenite (copper phosphate, not previously found in the United States), brochantite, alunite, gypsum, spangolite (basic chloro-sulphate of copper and aluminum), chalchanthite, goslarite, epsomite and gerhardtite (a basic copper nitrate forming green crusts on weathered surfaces of porphyry, and associated in these with a copper chloride, possibly atacamite).

The deposits with payable copper-ore take many widely differing forms, as follows:—

Deposits in limestone and shale, not connected with fissure-veins.
- Irregular bodies near contacts of main stock or dikes.
- Tabular bodies near contacts of main stock or dikes following stratification.
- Tabular bodies, following contacts of porphyry dike (all of these carry oxidized ores, almost exclusively; rarely chalcocite).

Fissure-veins.
- Normal veins in porphyry or in any of the other rocks near porphyry-contacts. Include central veins and surrounding partly-replaced porphyry forming together a lode. Carry chalcocite as the important ore; in upper levels also sometimes oxidized ores.
- Normal veins, following porphyry dikes in granite. Chalcocite and oxidized copper-ores.
- Normal veins following diabase-dikes. Chalcocite and oxidized copper-ores.

Stock-werks. Irregular disseminations in porphyry, quartzite and other rocks. Contain chalcocite and oxidized copper-ores.

The above classification is based on occurrence and form, and a more general genetic system, given below, shows a somewhat different arrangement.

Native copper, all of the oxy-salts of copper, and chalcocite are wholly secondary minerals produced by direct or indirect oxidation from primary pyritic ores. In all of the divisions given above, this primary ore consists of pyrite and chalcopyrite, with some zinc-blende and molybdenite. The scant gangue of the veins consists of quartz, while the deposits in

the first division are usually accompanied by garnet, epidote, magnetite, diopside, tremolite, or their products of oxidation.

Metamorphic Processes.

The region described in this paper is practically unaffected by regional or dynamic metamorphism, and even the ordinary hydro-metamorphism has altered the rocks but little, producing some slight changes in granite and porphyry and introducing cherts into the limestones. Epidote, chlorite, and a little pyrite are apt to develop in the hornblendic phases of the porphyry.

Contact-Metamorphism.

Contacts of porphyry with sedimentary rocks often show typical instances of this metamorphism. The granite-porphyry and the quartz-monzonite porphyry show themselves most effective in this direction, while there is usually but little metamorphic action at the contacts of the diorite-porphyry. The effect seems in direct proportion to the amount of quartz contained in the porphyry. Granite and quartzite are unaltered; the shales and sandstones of the Cretaceous series are hardened and baked. The shales change, as a rule, only at the immediate contact, to dense, greenish hornfels.

The Paleozoic limestone-series comes in contact with the main stock in two places—at Morenci and at Metcalf. In both places extensive copper-deposits are encountered. Dikes also occur at both places and along some of these radiating out into the unaltered areas the metamorphic processes may be examined to best advantage.

In studying the phenomena along dikes, it is found that the metamorphism varies greatly in the different strata, and even in apparently similar limestone layers there may be great difference in the degree of alteration. A well-defined dike 50 ft. wide on Modoc mountain was studied with special care as it cut through all of the formations present. Where contained in the Longfellow limestone the metamorphism extends at most 20 ft. outward into the limestone, and generally only a few feet. Garnet, epidote, diopside, specularite, and magnetite are the minerals which form abundantly by metasomatic replacement along the contacts, and intergrown with them are chalcopyrite, pyrite, and zinc-blende, unquestionably of contemporaneous

formation. The contact metamorphic limestone has certainly a very different composition from the unaltered rock, and it is apparent that much silica, iron, copper and zinc, at least, have been added. Epidote often forms in considerable quantities close to the contacts, while a little farther away garnet prevails. The Morenci shales overlying the Longfellow limestone are hardened and baked, but not materially altered in composition. Finally, when the dike enters the pure limestone of the Modoc formation, garnet forms in enormous quantities from the latter; the metamorphism exerted by the dike here merges into that affecting the whole block of limestone, due to the contact of the main stock of porphyry.

The principal metamorphic area at Morenci is about 2 miles long; its width is from 1,000 to 1,500 feet. The Modoc formation, however, has been affected to an extraordinary degree, and extends as a stratum of garnet and magnetite 2,000 ft. distant from the contact between almost unaltered Devonian and Cretaceous sediments.

The Longfellow limestone, though somewhat irregularly altered, is, next to the contact, generally transformed into a coarsely granular rock of garnet, epidote, diopside, magnetite, pyrite, zinc-blende, and chalcopyrite, and there is a most decided change of composition, chiefly consisting in additions of silica and iron. Other parts are less altered, but contain much magnetite, together with the sulphides mentioned, disseminated through a medium-grained crystalline mass of the carbonates of lime and magnesia. The development of magnetite, metasomatically, is a most pronounced feature of the process. Looking at the formation as a whole; sulphur, iron, copper and zinc have certainly been added, and probably also silica and magnesia. Large masses of magnetite, locally used as a flux, have been mined at this horizon.

The Morenci shales have suffered a change to dense, greenish hornfels with a development of amphibole and, in places, also muscovite. Pyrite and magnetite are also present, but on the whole, the change in composition is probably slight.

The Modoc limestone, containing about 96 per cent. of carbonate of lime, has proved extremely susceptible, and over large areas at Morenci, as well as at Metcalf, it is converted to a massive sheet of lime-iron garnet; magnetite is usually present;

epidote and oxidized copper-ores are also of frequent occurrence. This transformation, involving large additions of silica and ferric iron is very noteworthy, as the evidence is entirely uncontrovertible. Analyses of the garnet and of the unaltered limestone are given in my report, now in press.

The contacts of the sedimentary series and the porphyry are sharp and show no evidence of assimilation. All of the contact metamorphic rocks, when unaffected by oxidation, are very compact and hard, atmospheric waters finding great difficulty in attacking them. Considering that great amounts of carbon dioxide have been expelled during the metamorphism, it is clear that a great shrinkage of volume should have taken place if no additions of material had been received. Such a reduction of volume has evidently not taken place, and I believe that any loss has been fully balanced by gains from material contained in magmatic solutions.

The question whether contact metamorphic rocks simply represent a recrystallization, or whether they have received additional substance from the cooling magma, is a most important one. Prof. Rosenbusch believes that little or no additional substance has been received and considers that it is possible to determine the original character of metamorphic rocks from their present composition.[1] This idea has recently been followed out by Dr. J. Barrell in his study of certain contact metamorphic rocks of Montana. In this paper[2] he advances the following generalization that "carbonic acid is only expelled where the siliceous impurities of the limestone are sufficient to combine with the lime set free." Based on this he obtains the further result that a great loss of volume has taken place and that it is possible to calculate original constituents (kaolin, silica, magnesite, and calcite) from any given rock more or less altered to wollastonite, garnet, epidote, etc. If these statements are really meant as generalizations, as would appear from the paper, they are not supported by convincing proofs. Magmatic additions, though mentioned, are not considered important.

[1] Man kann es also als ein Gesetz aussprechen dass bei der Kontakt-metamorphose um Tiefengesteine das Eruptivgestein nur physikalisch und im allgemeinen nicht durch Stoffabgabe chemisch wirkte. *Mikroskopische Physiographie*, 3d ed., p. 85.

[2] The Physical Effects of Contact-Metamorphism, *American Journal of Science*, vol. xiii., pp. 279 to 296 (April, 1902).

Prof. Zirkel says[3] that in nearly all cases the contact-metamorphic rocks simply represent a recrystallization of original constituents. He believes that the contact-metamorphism simply took place by reason of the pressure and heat exerted by the molten rock, and speaks rather slightingly of the view that substance from the magma can be transferred to the surrounding strata, although admitting that in one or two cases it seems to have happened. Hawes, as is well known, proved—in the case of the Albany contact-zone—an increase of silica towards the contact and also a certain amount of boron which appeared to have been given off by the granite. Prof. Brögger, to whom we owe a most careful description of the Kristiania contact-zone, says, that on the whole, the alteration seems to involve chiefly a recrystallization, although certain of the phenomena strongly suggest local accession of material, though rather from adjacent strata than from the intrusive body.

This seems a rather crushing array of authoritative testimony from the petrographic side and it has even been intimated by Prof. Klockmann,[4] in a recent paper combatting the theory of transfer of material from magmas to sediments, that it ought to be sufficient to settle the question. While I do not doubt in the least the correctness of the conclusions drawn in individual cases by such eminent authors as Professors Rosenbusch and Zirkel, it is certain that contact-metamorphism manifests itself in many various ways, and that the particular phases connected with mineral deposits have been rather conspicuously neglected by many petrographers, whose data and statements in regard to the occurrence of ores, even in ordinary rocks, have always seemed to me to suffer somewhat from lack of detail and precision.

On the other hand, many French authors, among these Prof. Michel-Levy, and, lately, Prof. J. H. L. Vogt,[5] of Kristiania, together with a growing number of younger scientific men, have strongly contended that many substances are given off by the cooling magma and enter the adjoining strata. This view is

[3] *Lehrbuch der Petrographie*, 2d ed., vol. i., p. 587, 588.

[4] *Zeitschrift für Praktische Geologie*, vol. xii., p. 78 (March, 1904).

[5] See *Trans.*, xxxi., 637 to 640 and *The Genesis of Ore-Deposits*, p. 648, 2d ed. (1902).

shared by myself and expressed in a recent paper on contact-metamorphic deposits.[6]

The truth seems to be that in many cases no perceptible accession of substance from the magma has taken place, while perhaps, equally often, important additions have been received. How far the heat and the gases from the intruded magma will penetrate, and what effects they will produce, depend on many factors. As shown above, the composition of the magma is sometimes a factor of importance. In case of the Morenci contact-zone, the quantity of substance available seems to stand in direct relation to the quantity of quartz in the porphyry. In many intrusive magmas there may be a very small quantity of water present; the access of material may then be slight and the contact-phenomena mostly due to the heat of the rock. The difference in susceptibility of the various beds is also strongly marked; all investigators agree on this point. In this district impure and very compact limestones resist alteration much more than coarse-grained pure rocks of the same kind; and the change in composition in the case of clay-shales is extremely slight.

Study of the Morenci contact-zone, as a whole, proves conclusively that most important accessions of substance have been received. The rocks inside of the altered zone contain an enormous quantity of sulphur, iron, copper, and zinc. Iron is, of course, contained in the unaltered rocks to some slight extent, but in nothing like the quantities accumulated in the contact-zone; sulphur, copper and zinc in noticeable amounts are absent from the unaltered rocks. Were they present, to the extent of a small fraction of the percentage contained in the contact-zone, they could be detected, either directly or through the products of their surface-oxidation. The minerals in which these substances are contained were certainly formed contemporaneously with the ordinary contact-minerals of the district, like garnet, diopside and epidote.

The metasomatic development of magnetite in pure limestones which has recently been questioned by Prof. Klockmann may be observed in almost countless localities at Morenci and Metcalf, both in the field and under the microscope. It is

[6] *Trans.*, xxxi., 226–244 (1901) and *The Genesis of Ore-Deposits*, p. 716, 2d ed. (1902).

known that iron was not contained to this extent in the original rock, but to demonstrate its actual derivation in each case is, of course, difficult. The question becomes clear only when we compare the contact-zone as a whole with the original unaltered rocks.[7]

It seems very strange that any one can doubt the possibility of such additions, and overlook what must happen when a magma in aqueous fusion is suddenly brought up to higher levels and strongly ionized water-gas above the critical temperature is largely released from its bonds. It must of necessity contain dissolved substances. Even at comparatively low temperatures water is one of the most powerful solvents known, and its action, when a perfect gas, is probably far in excess of that at 100° or 200° C. It is well known that some rapidly congealed rocks, like the "pitch stone" from Saxony, contains up to 8 per cent. of water, indicating an amount of water-gas which, per cubic meter of magma, would at + 4° C. correspond to from 250 to 300 liters.[8] All magmas may, of course, not have contained this amount. The water-gas seems to have penetrated the limestones like a sponge, inducing extreme molecular mobility. Even should we deny any additions of substance, a most remarkable transferring of substance has certainly taken place in the rock, as shown, for instance, by large crystals of garnet developing in limestones of uniform

[7] Professor Klockmann's article, which is really intended to prove that no important deposits of magnetite can have a contact-metamorphic origin and that no important material can be transferred from cooling-magmas to adjoining sediments, was published in *Zeitschrift für Praktische Geologie*, vol. xii., p. 78 (1904). Among the arguments used is one referring to the content of alumina in epidote, and to the improbability of transfer of that metal from magma to limestone. At Morenci the massive epidote is usually confined to the immediate vicinity of the contacts, and I fully believe that some transfer of alumina has actually occurred. Regarding the occurrence of that mineral within the contact-metamorphic zone, but at some distance from the actual contact, it is not likely that the alumina represents an addition from the magma, but it is certain that under the peculiar conditions obtaining during the metamorphism the alumina to no small degree shared in that wonderful molecular mobility which characterized the whole process.

When, however, Prof. Klockmann refers to garnet as an essentially aluminous mineral, a typographical error must surely have occurred. All garnets do not contain alumina, and the contact-metamorphic garnets at Morenci and Metcalf are throughout andradites or lime-iron garnets.

[8] E. Weinschenk, Vergleichende Studien ueber den Kontakt-Metamorphismus, *Zeitschrift der deutschen Geologischen Gesellschaft*, vol. liv., p. 443 (1902).

composition containing far less iron and silica than the amounts required by the newly-formed mineral. In a chapter on inclusions cogent proof will be brought that the magma actually was accompanied by water that contained a large amount of substances in solution.

A misapprehension of the character of contact-deposits seems to exist in many quarters, as shown, for instance, by Prof. Klockmann,[9] in considering the presence of minerals containing boron, fluorine, etc., as necessary to prove the contact-metamorphic character of these deposits. To such arguments I would reply that the character of magmatic waters evidently varies greatly in different magmas. Some may carry large quantities of the substances mentioned, as shown by the presence of tourmaline, cassiterite and other minerals in the contact-metamorphic rocks, while others may be almost wholly deficient in them, and, instead, carry sulphur, copper, iron, and other metals. In the Clifton group of deposits, I would be inclined to consider molybdenum a characteristic constituent, taking the place of tungsten in the tin-deposits. Any attempt to reduce the wonderful variety in the contact-metamorphic deposits to a single pattern is doomed to failure.

In a short paper dealing with contact-metamorphic deposits in North America,[10] I emphasized the irregular form of most ore-deposits of this kind and declared that they only occur close to the contact or, at most, a hundred feet away. As a result of wider observations I would modify this statement; as far as we know at present, they may occur several hundred or even 2,000 ft. away from the contact. In fact, disseminated sulphides and magnetite occur at Morenci as far as 2,000 ft. from the main contact.

A tabular form of deposits often noted is usually due to the strongly marked difference in susceptibility of the various beds. Wherever the deposits have been enriched by oxidation the form may be more or less dependent upon these changes.

Mr. W. H. Weed has noted the tabular shape in contact-deposits at Cananea, Mexico, and makes the form a basis of classification.[11] I do not believe, however, that distance from

[9] *Zeitschrift für Praktische Geologie*, vol. xii., p. 75.

[10] *Trans.*, xxxi., 226 (1901).

[11] Ore-Deposits near Igneous-Contacts, *Trans.*, xxxiii., 715–746 (1903).

contact (within limits given above) and shape are at all essential, and can find no mineralogical difference between deposits varying in these respects.

Hydrothermal Metamorphism.

Both the porphyry- and the contact-zone are traversed by fissure-veins which carry pyrite with a small quantity of chalcopyrite, zinc-blende and molybdenite, whenever oxidation has not changed these minerals. These pyritic veins are of low—generally unpayable—tenor; they consist of prevailingly granular or massive, very rarely crustified, minerals with a little quartz-gangue, and are believed to have been formed by pyritic replacement along well-defined fissure-planes. In the porphyry these veins are surrounded by very wide zones in which the rock is greatly altered by the introduction of sericite and pyrite, and this applies to Metcalf as well as to Morenci. At the latter place almost the whole of Copper mountain, containing the most important lodes, is thus altered. The process which, in my complete report, is elucidated by many analyses, produces bleached rocks of varying hardness in which all of the feldspar has been replaced by sericite and some pyrite. The biotite and hornblende are transformed into chlorite and serpentine, while the silica of the rock remains almost constant. All of the lime and soda is eliminated, while potash is greatly increased. No carbonates are formed during this process.

Little alteration is noted where fissure-veins cut through contact-metamorphic shale, nor when highly altered garnet-magnetite rock forms the walls, but in unaltered or slightly metamorphosed limestone a change is observed, For a short distance from the vein—a few inches or a few feet—the rock is bleached and proves to have been converted into tremolite, more rarely diopside, with disseminated pyrite, chalcopyrite, and zinc-blende, all more or less intimately intergrown with magnetite. This alteration involves a loss of carbon dioxide and some lime, as well as addition of silica, iron, magnesia, and the sulphides mentioned above. More rarely argillaceous limestones are altered to sericitic minerals with magnetite and sulphides. Magnetite has not been observed in the massive pyritic veins, although it occurs in the altered country-rock.

Relation of Contact and Hydrothermal Metamorphism.

The alteration of limestone along fissure-veins to tremolite (or diopside) with magnetite and sulphide is, so far as I know, a novel one. Ordinarily, limestone alters next to fissure-veins to dolomite or quartz or jasperoids.[12] The addition of silica and iron and the mineralogical trend of the hydrothermal process at Morenci undoubtedly connects it in some way with contact-metamorphism, making it probable that the alteration took place at high temperature comparatively soon after the solidification of the porphyry.

At first glance, it might seem plausible to assign all the changes which have taken place in the metamorphic zone to the same hydrothermal alteration which has affected the porphyry along the fissure-veins. This view, however tempting, is surely incorrect. Instead of one set of phenomena, there are two related and, in part, superimposed processes. Among the proofs of this are the absence of sericitization in the porphyry of many dikes which have exerted strong contact-metamorphism. Further, the entire independence which the masses of extremely altered garnet-magnetite rock show in regard to the fissure-veins, and their distinct dependence on contact of dikes and main stock. The vein-alteration produces dull, earthy rocks from the limestone, while contact-metamorphism results in hard, compact and granular rocks. On the other hand, there is undoubtedly a certain relation between the two processes for amphibole (and pyroxene) occurs in the true contact-metamorphic rocks, and the intergrowth of magnetite and pyrite is characteristic of both. I should, therefore, conclude that after the completion of the contact-metamorphism, properly speaking, and after the consolidation of the porphyry, an extensive fissuring took place and solutions derived from the cooling porphyry, probably ascending and laterally extending from this rock, flowed through these fissures. Everything indicates that these solutions were closely related to those which emanated from the magma at the moment of intrusion and, in fact, similar in general composition.

As to the quantitative relation of contact-metamorphism and hydrothermal metamorphism, it is difficult to speak with ab-

[12] W. Lindgren, Metasomatic Processes in Fissure-Veins, *Trans.*, xxx., 578 to 692.

solute certainty. In some parts of the contact-metamorphic rocks small seams with sulphides and magnetite are very abundant (for instance, in the Yavapai mine) and it is not easy to say how much of this should be credited to each form of alteration, for the sedimentary rocks were evidently solid during the intrusion, and seams filled with sulphides may well have formed in them. Generally speaking, they would be, and, in fact, are at present much less permeable to solutions than the porphyry, as shown by the slight depth which oxidation has attained in them, and it is believed that the hydrothermal solutions were chiefly confined to cracks and fissures in contradistinction to the much more searching action of gaseous water. The facts above given show indeed how slight is the lateral spread of alteration from the veins in limestone. That the solutions producing the contact-metamorphism effected such intense results is probably due to the existence of a far greater degree of heat and gas-pressure.

Processes Due to Oxidation and Hydration in the Altered Zone.

Under influence of surface-waters (but protected from direct oxidation), chlorite and serpentine form from the tremolite and diopside of the contact-zones, while garnet is little altered. Under direct oxidizing-action, garnet changes to limonite and quartz, while lime is probably carried away as carbonate. Tremolite and related minerals are similarly affected. Magnetite oxidizes to hematite and limonite, many large bodies of which are mined for fluxing-purposes. Pyrite changes by direct oxidation into ferrous sulphate and free sulphuric acid; the ferrous sulphate upon further oxidation yields ferric sulphate and the latter is easily decomposed into basic sulphates, ferric hydrates and free acid; ferric sulphate is also ready to attack pyrite and other sulphides, changing them to sulphates and being itself reduced to ferrous sulphate.

This cycle of reactions will finally transform all of the sulphides present into various, more or less soluble, oxy-salts. A large part of the pyrite will be changed to limonite. Such "iron caps" are seen at the outcrops of many veins in regions where oxidation proceeds undisturbed by erosion, but in this region they are generally absent.

The veins are marked by siliceous outcrops, either entirely

barren or containing small amounts of oxidized copper-ores. No basic ferric sulphates have been seen either in the mines or on the surface. The waters percolating downwards must soon have lost their oxygen and, for the alteration at greater depths, ferrous sulphate and diluted sulphuric acid are probably the only reagents of importance resulting from the pyrite. It is clear that the upper part of the veins have not been formed by simple oxidation of pyrite and accumulation of limonite.

Direct oxidation of chalcopyrite yields cupric and ferrous sulphates, while the zinc-blende produces zinc sulphate; both of these processes are in evidence wherever the disseminated sulphides in the metamorphic rocks are exposed to oxidation. The general order of attack of oxygen is usually stated as follows:—Arsenopyrite (most easily attacked), pyrite, chalcopyrite, blende, galena and, finally, chalcocite (most difficultly attacked). This rule is probably true only for one set of conditions as to mass, aggregate and character of solutions. Very marked exceptions from it occur at Morenci.

Sulphate Waters.—Descending-waters from a lode of decomposing pyrite, chalcopyrite and zinc-blende should contain chiefly ferrous, cupric and zinc sulphates, together with free sulphuric acid. The first two salts are easily soluble, but far more so is the zinc sulphate.

Cuprous sulphate is stable only under certain conditions and is not believed to be an important reagent, though it may form during subsidiary or intermediate reactions. Its presence in any mine-waters has not been satisfactorily proved.

In this district, some moisture percolates the upper workings, as shown by efflorescences and stalactites of sulphates, but the mines are practically dry. In the porous porphyry the moisture spreads easily, while the hard metamorphic rocks offer considerable resistance.

Processes in Fissure-Veins.—Below the region of oxidizing-influences the veins consist of pyrite, chalcopyrite and zinc-blende, while the upper few hundred feet contain chalcocite and oxidized copper-ores. The most important action of the descending sulphate solutions has been a development of chalcocite by the action of cupric sulphate on primary pyrite-ore; this process began at the time the veins were first touched by

oxidizing waters, and continues to some extent to the present time.

If secondary deposition of sulphides has taken place according to Schuermann's reactions, they thould be arranged in the upper zones of the deposit in the following order:—(*Top*), galena, zinc-blende, chalcocite, bornite, chalcopyrite and pyrite. (*Bottom.*)

In Morenci, practically only chalcocite thus forms. The first of the primary minerals attacked is the zinc-blende which appears to be rapidly transformed, first into covellite and then into chalcocite, as follows:

$$ZnS + CuSO_4 = ZnSO_4 + CuS.$$

Chalcopyrite, present only in small quantities, is probably attacked at the same time. The zinc sulphate is carried away and no zinc minerals appear in the upper, oxidized part of the veins.

Contrary to the list just given above, blende is thus attacked before the pyrite. The conversion of pyrite to chalcocite may be studied in all stages of the process; it is a molecular replacement attacking the pyrite from cracks and fissures, and gradually eliminating it entirely. However, even in the best chalcocite-ores residual pyritic cores ordinarily remain. Dr. Stokes has shown that the reaction at + 100° C. and + 200° C. proceeds as follows:

$5\,FeS_2 + 14\,CuSO_4 + 12\,H_2O = 7\,Cu_2S + 5\,FeSO_4 + 9\,H_2SO_4 + 3\,H_2SO_4$, the last H_2SO_4 being formed by oxidation of the sulphur of FeS_2. It is probable that the reaction likewise goes on, though more slowly, between + 100° C. and + 20° C., the range probably existing in the deposit during the period of oxidation. Mr. H. V. Winchell's reaction[13] necessitates sulphurous acid as a reagent, the presence of which seems unlikely. The equations given by Prof. Van Hise[14] for the formation of secondary copper sulphides seem improbable, as they require

[13] Synthesis of Chalcocite and Its Genesis at Butte, Montana, *Bulletin of the Geological Society of America*, vol. xiv., pp. 269-276.

[14] Some Principals Controlling the Deposition of Ores, *Trans.*, xxx., 101, 111, 112.

the presence of free oxygen, and as they are generally based on cuprous sulphate.

By this process of alteration the massive pyritic veins have been transformed into almost solid, dull, sooty chalcocite; and the zones of dissemination surrounding them in the porphyry have changed into low-grade chalcocite-ores. The process is accompanied by the formation of some kaolin, quartz, chalcedony, and probably also opal. No sericite forms. The kaolin develops metasomatically at the expense of the sericite, and sulphuric acid was evidently the reagent. Alunite is in places formed in the same manner. Extremely large amounts of ferrous sulphate must have been carried away during this process.

Oxidation of Chalcocite.—After a large part of the chalcocite in the lode had been formed, there came a time when erosion and degradation, working faster than oxidation, began to expose the upper part of the chalcocite-zone to active and direct attack by oxygen. Practically all of the veins are in this stage. The present zones of active oxidation are therefore due, not to direct oxidation of the primary lode, but to the destruction of the upper horizon of the chalcocite-zone. As reagents, there are here ferrous sulphate, sulphuric acid, cupric sulphate and free oxygen. Any ferric sulphate present would soon be reduced to ferrous salt by the abundant pyrite. Chalcocite changes to cuprite and cupric sulphate, sometimes with an intermediate stage of covellite.

$$2\,Cu_2S + O = 2\,CuS + Cu_2O \text{ and } CuS + 4O = CuSO_4.$$

Cuprite partly reduced to metallic copper is, in fact, abundantly present in the upper limit of the chalcocite-zone.

$$Cu_2O + H_2SO_4 = Cu + CuSO_4 + H_2O.$$

By some process not quite elucidated, chalcocite may, in places, change directly into native copper. The zone of cuprite and metallic copper has rarely great vertical extension, for the reason that both minerals are rapidly converted into cupric sulphate, brochantite, malachite, azurite and chrysocolla; but these products are soon dissolved by free sulphuric acid from pyrite, a mineral which tenaciously remains until the last, and descend

as soluble sulphates to enrich the upper part of the chalcocite zone.

In the Copper Mountain veins, contained in porphyry, oxidized copper-minerals are almost entirely absent, probably because of the great amount of free sulphuric acid present. The chalcocite disappears at from 100 or 200 ft. below the surface, and the lode consists of a rusty mass of sericitized porphyry, with considerable amounts of remaining pyrite and efflorescences of cupric sulphate. The last particles of pyrite only disappear close to the surface.

In places, chalcocite may change directly into brochantite or malachite. Cupric oxide (melaconite or tenorite) has not been found.

Oxidizing Processes in Limestone.—The processes which have resulted in the oxidized deposits now forming irregular or tabular masses in limestone or shale, and not connected with fissures, are materially different from those occurring in the lodes. In most cases the original material consisted of disseminated lean pyritic ores, containing pyrite, chalcopyrite, zinc-blende, and magnetite, locally more or less concentrated in irregular masses, or following certain strata- or dike-contacts. Free oxygen and sulphuric acid, ferrous and cupric sulphates, with an abundance of calcium carbonate, formed the reagents. Though oxygen and carbon dioxide may in part have produced limonite and malachite directly from pyrite and chalcopyrite, the most important reactions doubtless were those between calcium carbonate and sulphate solutions, partly derived from nearer the surface.

$$2\ CuSO_4 + 2\ CaCO_3 + H_2O = (CuOH)_2, CO_3, + 2\ CaSO_4 + CO_2 + H_2SO_4.$$

$$3\ CuSO_4 + 3\ CaCO_3 + H_2O = (CuOH)_2, Cu(CO_3)_2 + 3\ CaSO_4 + CO_2.$$

In the first case malachite, in the second azurite forms, together with gypsum. The latter mineral, though largely carried away in solutions, is not uncommon in these deposits at Morenci. Ferric hydrate will be produced from ferrous sulphate and calcium carbonate. Thus, in general, is explained the constant occurrence in these deposits of malachite, azurite

and limonite. Crusts of replacing-malachite and azurite frequently surround residual, rounded masses of limestone. The gangue of garnet and magnetite alters to ferric hydrate and quartz. Chrysocolla forms when silica is plentiful; zinc-minerals are not uncommon in these deposits, and efflorescences of the zinc and magnesia sulphates sometimes cover the tunnel-walls. During the process outlined above, lime is almost wholly eliminated, while alumina forms residual concentrations.

Oxidizing Processes in Shale.—Disseminated cuprite in beds of Morenci shale is a common occurrence, and some of the large ore-bodies of the Manganese Blue mine were of this character. It occurs as flakes on the bedding-planes, or in small replacement-veins, sometimes accompanied by distinctly-later malachite and by limonite. These occurrences seemed difficult to explain, but light is thrown on them by recent experiments by Dr. E. Kohler,[15] who shows that solutions of cupric sulphate, filtered through kaolin, become hydrolyzed by adsorption. The copper is precipitated as oxide, and sulphuric acid is set free. (Experiments are now in progress which seem to indicate that the Morenci shale possesses remarkable power of adsorbing copper from aqueous solutions.)

Azurite also occurs frequently in shale, as shown by the second ore-body in the Detroit mine. Large crystals of that mineral develop here, metasomatically, in a metamorphic shale composed of sericite and amphibole. Cases have been observed where azurite envelops masses of undecomposed pyrite accompanied by a little limonite. During the oxidizing process the alumina possesses considerable mobility. It is dissolved by sulphuric acid from clay-shale and forms certain aluminous minerals, notably sericite. The aluminium sulphate formed is extremely soluble in water, and thus easily transported. At many places in the mines of Morenci, pure kaolin forms, together with azurite and malachite.

Paragenesis.

The minerals formed during successive stages are as follows:

[15] Adsorptionsprozesse als Faktoren der Lagerstättenbildung und Lithogenesis, *Zeitschrift für praktische Geologie*, vol. xi., p. 49 (1903).

VEINS.	
Primary Processes.	Pyrite, chalcopyrite, zinc-blende, molybdenite (rarely magnetite), sericite, quartz, rarely tremolite, diopside and epidote. Introduction of iron, sulphur, zinc, copper, molybdenum, potassium and silica. Elimination of calcium and sodium.
Processes of Oxidation. Action of sulphate solutions without oxygen.	Chalcocite (rarely covellite, chalcopyrite, and bornite), quartz, chalcedony, kaolin, alunite. Introduction of copper. Elimination of zinc, some iron and sulphur.
Action of directly oxidizing solutions.	Cuprite, native copper, brochantite, malachite, chrysocolla (rarely azurite), chalchanthite, limonite, quartz. Introduction of carbon dioxide. Elimination of sulphur, together with some iron and copper.
CONTACT-DEPOSITS.	
Primary Processes.	Pyrite, magnetite, chalcopyrite, zinc-blende, molybdenite, specularite, garnet, epidote, diopside, tremolite, quartz. Introduction of much iron and silica, together with copper, zinc, molybdenum, sulphur, possibly magnesia. Elimination of carbon dioxide and probably some lime.
Processes of Oxidation.	Limonite, malachite, azurite, cuprite, rarely native copper and chalcocite, copper-pitch ore, chrysocolla, goslarite, zinc carbonate, willemite, calamine, pyrolusite, quartz, calcite, chlorite, serpentine. Introduction of carbon dioxide and additional copper. Elimination of calcium, sulphur, some zinc and iron.

CHARACTERISTICS OF DEPOSITS.

Deposits of Carbonates and Oxides in Limestone and Shale.

The important occurrences of these ore-bodies are found in the Longfellow, Manganese Blue, Detroit, Copper Mountain, Montezuma, and Shannon mines.

They contain practically all of the oxy-salts of copper known from the district; but chiefly malachite, azurite and cuprite, with very subordinate amounts of native copper and chalcocite. The accompanying minerals consist of limonite, manganese peroxide, kaolin, and soft, yellowish material which, in a large proportion of deposits, generally proves to be decompos-

ing and hydrated forms of tremolite, diopside, garnet, or epidote. Some deposits of chrysocolla and limonite occur, however, in unaltered limestone, and the cuprite-bearing shales are not accompanied by any gangue except a little limonite. The copper-minerals are sometimes formed by crustification, but replacements in shale or lime are almost equally common.

The form is sometimes wholly irregular, but there is a marked tendency towards a tabular form following certain strata evidently favorable for deposition. The horizontal extent varies greatly, but rarely exceeds 300 or 400 ft., and the two dimensions are apt to be approximately equal. The thickness ranges from 1 to 30 ft. and is sometimes more. Regular boundaries rarely occur, and the pay-ore easily shades off into the country-rock. In the Manganese Blue and the Detroit mines several superimposed ore-bearing horizons are found within 300 ft. of the surface. The Longfellow deposit has the form of an inverted pyramid.

Acidic porphyry is found in the immediate vicinity of all deposits. Some of them immediately adjoin the contact of the main stock, but others show decided dependence upon dikes of porphyry; one class of deposits forms tabular masses along the contacts of dikes; others, such as the Detroit, the Manganese Blue, and the Longfellow, lie between two or three porphyry dikes, the latter being largely barren, except for some disseminated chalcocite. Fissures antedating the oxidation, but subsequent to the contact-metamorphism, have sometimes influenced the form by guiding the descending waters.

The deposits may be found in any of the Paleozoic horizons between the Coronado quartzite and the Cretaceous beds. They frequently crop at the surface, azurite appearing to resist decomposition quite obstinately.

Driving laterally, or sinking deeper from these ore-bodies, is apt to develop hard limestone with typical contact-metamorphic minerals and scattered pyritic ores. The very confident conclusion has been drawn that the majority of these deposits have been formed by the oxidation of irregular or tabular masses of low-grade pyritic ores, such as the lower mine-workings have disclosed in such abundance, for instance, in the Yavapai mine.

An enrichment accompanied the oxidation, both on account

of reduction in volume and introduction of additional copper as sulphate. The oxidation does not reach the water-level which is far below the present workings, but acts most irregularly, sometimes leaving fresh metamorphic limestone near the surface and again penetrating along fissures to a depth of 400 feet.

The above explanation does not fit all of these deposits, for some are unaccompanied by evidence of contact-metamorphism and have probably primarily been formed by the action of thermal, vein-forming waters, and others are entirely of secondary origin, being deposited in shales and in the clays along important faults by migrating sulphate solutions. The oxidation of the present ore-bodies followed the faulting movement and probably began at a rather early epoch of Tertiary times.

Fissure-Veins and Related Deposits of Morenci Type.

Fissure-veins with a northeasterly or northerly trend, but of no great individual length, follow the entire length of the porphyry stock, but are especially developed between Morenci and Metcalf and on Copper King mountain. The most prominent lode system at Morenci extends for about 2 miles, and consists of a number of shorter-linked and branching fissures, forming two belts slightly curved towards the southeast. One of them lies in porphyry within a few hundred feet of the contact and comprises the principal mines of the district—the West Yankie, the Humboldt, and the Copper Mountain. The other and parallel system traverses the metamorphic rocks a few hundred feet southeast of the contact.

The dip is steep to the NW. or SE. and the system is thus a conjugated one, bearing every evidence of origin by compressive stress.

Outcrops are very poor, frequently wholly unrecognizable, and it is most difficult to trace the veins on the surface. Low-grade malachite, chrysocolla, and brochantite-ores are contained in the outcrops of some veins. Large masses of limonite while common enough in the oxidized contact-metamorphic deposits do not usually occur in the vein-croppings.

Underground exposures always show one or more well-defined walls frequently polished and striated. The faulting-movement on these fissures is slight.

In the majority of these deposits there is a central vein which ordinarily is 4 ft. wide or less, but may sometimes expand to 50 ft.; it is composed of nearly massive sulphides, and closely follows the fissure. In the porphyry, this is surrounded by a much wider zone of altered rock which may constitute ore; the central vein and the altered zone together are characterized as a "lode." In almost all of the veins the following vertical distribution of ores is noted:

Surface Zone.	From 50 to 200 ft. deep from the croppings. Contains oxidized copper-minerals or is barren.
Chalcocite Zone.	From 100 to 400 ft. in vertical extent; possibly more in places. Contains chalcocite and pyrite.
Pyritic Zone.	Begins from 200 to 600 ft. below the surface. Contains pyrite, chalcopyrite and zinc-blende.

It has been shown that the minerals of the two upper zones have been derived from those of the pyritic zone by processes of direct and indirect oxidation; that the chalcocite is wholly formed by replacement of pyrite effected by solutions of cupric sulphate; that secondary covellite, chalcopyrite and bornite only occur in very small amounts. Also, that the surface-zone is derived from the chalcocite-zone by its direct oxidation. It has further been emphasized that the pyritic zone is generally very poor; that the chalcocite-zone produces the richest ore, and that the surface-zone is always poor and sometimes barren.

The pyritic part of the veins is, with good reason, believed to represent the primary deposition of sulphides along the fissures. It contains a small amount of quartz-gangue with intimately intergrown pyrite, zinc-blende and chalcopyrite; the two last-named minerals are present only in small quantities; molybdenite also occurs. A well-defined foot-wall is usually present, while the hanging-wall may be more or less indistinct or represented by several subordinate fissures. The deposition seems to have been chiefly effected by metasomatic replacement of crushed and sheeted porphyry, or metamorphic rock. The zone of altered rock surrounding the vein consists of sericitized and pyritic porphyry when the vein is in this rock, and may then be very wide. In the hard, metamorphic limestone and shale, this zone is narrow and shows either an amphibolitic or a sericitic alteration and contains besides pyrite, chal-

copyrite and zinc-blende intergrown with magnetite, a mineral absent from the vein proper. In the chalcocite-zone, commercially the most important, magnetite has largely, but rarely wholly, replaced the pyrite, both in the massive veins and in the zones of pyritization and sericitization surrounding them at a certain depth. It is generally sharply separated from the pyritic zone, the transition taking place within a surprisingly short distance. Below this limit, evidence of chalcocitization can only be found along fissure-planes. The uppermost part of the chalcocite-zone is invariably the richest. The massive pyrite veins form pure, dull black chalcocite, or a mixture of pyrite and chalcocite. This constitutes high-grade ore with more than 5 per cent. of copper—ranging up to 70 per cent.—while the pyritized porphyry turns into low-grade ores with from 2 to 5 per cent. of copper.

To these large bodies of low-grade ore the recent great development of the district is due. In some mines, which are generally on the lower slopes, or in the bottoms of canyons, the chalcocite begins almost at the surface. At Morenci, situated high up on the hills, the depth from the surface is rarely less than 200 ft. The depth on the vein attained by the chalcocite-zone, from the level where direct oxidation begins to the upper limit of the pyrite-zone, varies greatly; it is sometimes less than 100 ft., while under Copper mountain the average would somewhat exceed 200 ft. Directly below the summit it is 300 ft., and its lower limit in some cases has not yet been reached. In general, the upper limit would be represented by a curved line somewhat less convex than the contours of the mountain. The lower limit seems to be flatter, but great irregularities exist, due no doubt to exceptional conditions of circulation of surface-waters.

The great bodies of low-grade ore are almost wholly conconfined to the lodes in porphyry; and the pay-zone generally contracts greatly when contact-metamorphic shales or limestones are entered. Stopes of low-grade ore range from a few feet to 100 ft. or more in width; many are 200 ft. long and have been stoped for the same vertical distance. The great body of concentrating ore between the two Humboldt walls, which dip against each other, is about 300 ft. long, up to 200 ft. wide, and has been stoped 200 ft. high. Values gradually

decrease on both sides unless, as sometimes happens, cut off by fissures on one or both sides. The limit is thus usually simply determined by the lowest grade of ore workable. Sericitization and chalcocitization has spread considerably farther, laterally, than is indicated by the 2.5 or 3 per cent. of copper contained in the lowest grade of ore mined.

The ore-shoots of Copper mountain are thus materially different from those usually found in gold- and silver-veins. Their greatest extent is horizontal rather than vertical, and their size depends more upon conditions of circulation of surface-waters than upon the primary distribution of copper in the vein. Prospecting for reserves must proceed laterally rather than towards extreme depth.

The surface-zone is always richest near the lowest limit of oxidation where cuprite and native copper form from chalcocite. The upper part contains poorer ores of malachite, brochantite, etc., and may be entirely barren.

In the Copper Mountain veins the chalcocite apparently changes directly to cupric sulphate; and other copper oxy-salts are practically absent. The pyrite remains, in part, rusty and accompanied by limonite, until near the surface, where it finally disappears. The surface-zone is thus directly derived from the chalcocite-zone by gradual erosion and oxidation, indicating that the latter is not a very recent development.

Two of the principal faults of the district cut across the Morenci veins and dislocate them. Some of the ore-bodies are clearly faulted, so that rich chalcocite-ore is brought opposite leached and barren surface-rock. Brecciated zones, accompanying this faulting, contain fragments of chalcocite-ore. In all, the evidence is pretty clear that, at least a part of the chalcocite-zone had already been formed when the faulting took place, and that, consequently, the beginning of chalcocitization and oxidation must be placed in the earlier or middle part of Tertiary times.

Descriptions of chalcocite-ores from other districts show that the secondary sulphides develop at a point just below the water-level. In none of the important mines in this district has the water-level been reached; it is probably far below the present workings. Chalcocite may now form in the upper part of the zone, in places away from fissures and faults, where

sulphate solutions are slowly descending and free oxygen scarce, but I doubt very much whether it is now forming in the lower part of that zone. I regard the chalcocite-zone as formed about an ancient, gradually-receding water-level. During the early Quaternary, that level was evidently several hundred feet higher than at present, but the occurrences of the chalcocite appear to require a still higher stand, such as existed during the probably moist climate of the middle Tertiary.

The payable deposits, as a rule, lie at high elevations, and no large deposits have been opened on the lower slopes of Chase Creek canyon, or along its bottom. The lower down on the slope a deposit is located, the nearer to the surface will the chalcocite-zone be found. Chalcocite-ores do occur, in fact, in veins along the bottom of Chase Creek canyon, but the mineral shows only as a slight coating on pyrite. This topographical distribution is the more remarkable when it is recalled that Chase Creek canyon antedates the early Quaternary conglomerates (Gila formation) and that, therefore, oxidation would apparently have had a long time in which to act. It confirms the conclusions as to the great age of the chalcocite-zone and emphasizes the very slow rate at which oxidation works.

Chalcocite-ores and oxidized ores forming a "stock-werk" of seams in porphyry (at Metcalf mines), or in quartzite (at the East Yankee mine), or occurring as disseminations in porphyry dikes (West Yankee lode and Shannon mine), in general correspond to the descriptions of the altered zones surrounding the veins. They pre-suppose an earlier sericitization and pyritization effected by the primary vein-forming solutions.

The fault-planes of the principal epoch of dislocations are later than the Morenci veins and generally barren, but may locally contain "drag" or cuprite-ores deposited from migrating sulphate solutions by processes of adsorption.

The Coronado Type of Veins.

Almost the only representative of the Coronado type of veins is the Coronado lode, which outcrops on the summit of Coronado ridge about 2,000 ft. above Metcalf. It presents the feature unusual for a fissure-vein of following one of the principal faults of the district with a throw ranging from 1,000 to 2,000 ft., and it is traceable for nearly 2 miles, finally disap-

pearing towards the west under the basaltic foothills of Eagle creek. The fault-zone, which is from 50 to 200 ft. wide, is followed by a diabase dike which has been crushed and partly deformed. Oxidized ores, malachite, azurite and chrysocolla, of medium to poor grade, occupy irregular shoots in the surface-zone, but are replaced at a depth of from 200 to 300 ft. by chalcocite-ores. At some points the latter, however, reach the surface. Explorations during the last two years are reported to have developed large bodies of these, even at a depth of 500 ft. below the surface. The evidence—as far as it goes—here, too, points to secondary origin of chalcocite and its derivation from pyritic ores, but here, as well as at Morenci, the maximum depth attained by the secondary chalcocite has not been fully demonstrated.

The Coronado lode was formed later than the Morenci type of veins and subsequently to the intrusion of diabase, which is younger than the porphyries of Morenci and Metcalf. Regarding the relative age of the diabase and the fault-fissure, the evidence is hardly conclusive. It seems as if the dike had been intruded during the epoch of faulting, and the solutions depositing the copper certainly followed the intrusion of the dike.

Gold-Bearing Veins.

Gold, silver and lead are practically absent from the principal deposits, but it is an interesting fact that they begin to appear in many of the outlying-veins somewhat distant from the central mass of porphyry. These veins, in which copper is apt to play a less important part, have not as yet attained much importance from an economic standpoint.

Conditions of Ground-Water.

Permanent water has not thus far been encountered in any of the mines in the whole district with which this paper deals. Morenci is situated on the hills from 800 to 1,500 ft. above the principal streams, Chase creek and Eagle creek; and the deepest workings in no place reach farther than 600 ft. below the surface. A little seepage from the surface takes place in case of heavy rains, or from the local water-supply; and some drifts and cross-cuts underneath the town are somewhat damp, especially in the Manganese Blue and Arizona Central mines. The mines at

Metcalf are situated on Shannon mountain from 500 to 1,200 ft. above Chase creek, and here, too, the workings are dry, excepting one place in the Shirley tunnel where a winze struck some standing water. The few shafts and prospects, sunk in the bottom of Chase creek, are the only places containing permanent water. The Copper King mine, situated a few hundred feet below the summit of the mountain of the same name, has a shaft 600 ft. deep; some crevices containing water have been found in it, but they soon drained out and no more has since come in.

The present stand of the water-level, except along the creeks, is practically unknown. It probably rises as a slightly curved surface from the creek-levels towards the high hills. The total amount of water stored below this water-level is probably small.

Depth of Oxidized Zone.

The presence of products of direct or indirect oxidation shows the depth to which the oxidizing-waters or the sulphate solutions have penetrated; but the porphyry and the metamorphosed limestones should be separated, because they act very differently. In that part of Copper mountain which has been explored, the average depth of the lower limit of the chalcocite-zone is 400 ft., but it increases in places to 500 or even 600 ft. To this depth from the surface, the sulphate solutions descended, and along important fissures they may have gone somewhat farther. The solutions not only followed fissures, but penetrated the porous, sericitized porphyry with considerable ease. On the other hand, the altered limestones and shales are very compact, non-porous and impervious. Where circulation was facilitated by fissures, as in the Manganese Blue and the Joy mines, the rocks may be partly oxidized to a depth of 400 ft., but this is generally a maximum. There is no well-defined plane expressing the depth of oxidation, which, on the contrary, proceeds very capriciously, fresh sulphides being frequently found very close to the surface.

Fluid-Inclusions.

Fluid-inclusions have been observed in the quartz-grains of granite, quartzite, porphyry and vein-quartz occurring in this district. There is nothing uncommon in this; it is, indeed, the

ordinary condition of affairs. As these fluid-inclusions, beyond doubt, contain aqueous solutions, it may be regarded as certain that such fluids were present when the quartz-grains in question were formed. It is the purpose of the following paragraphs to call attention to certain phenomena in these inclusions which have not been described previously.

In Granite.—Inclusions, filled with a fluid and a bubble of gas, occur abundantly in the quartz-grains of all of the granites examined. They are of rare occurrence, though not unknown in the feldspars. The form is round or elliptical, or may be that of a negative quartz-crystal with prism and pyramid. The size averages perhaps 0.02 mm. The relation of volume between the fluid and the bubble varies considerably among inclusions in the same grain. In the smaller inclusions the bubble frequently is in active motion. Heated to 40° or 50° C. there is no perceptible change in volume of fluid or bubble, and consequently it may be considered certain, that the fluid is not liquid carbon dioxide, which has sometimes been observed in the granites, but chiefly water. In some, but by no means all, of the inclusions there is also a solid body contained in the fluid; in some cases this is a transparent cube, in others, and more commonly, an irregular grain or a rod of the solid material. This also has occasionally been observed and described in granites from other districts.

In Metamorphic Limestones.—The metamorphism of the limestone to garnet, epidote, diopside, quartz and other minerals took place under conditions of high temperature and pressure, and almost certainly in the presence of aqueous solutions in fluid or gaseous form. It is a curious fact that these minerals only very exceptionally contain fluid-inclusions. The quartz-grains formed seem entirely homogeneous and free from these interpositions. Only one or two very small inclusions with moving bubble were noted. The same applies to the garnet and other heavy minerals. The calcite would hardly be expected to preserve any such inclusions on account of its perfect cleavage.

In Porphyry.—The inclusions in the porphyry are again practically confined to the quartz. They occur chiefly in the phenocrysts, but also in the quartz of the ground-mass, although they are here usually very small. Naturally, the diorite-porphyries

are poor in inclusions, but they appear abundantly in the granite-porphyries and the quartz-monzonite porphyries with which the ore-deposits are chiefly connected. They are found throughout the Metcalf granite-porphyry, which is characterized by large bi-pyramidal quartz-phenocrysts, and they are almost equally common in the smaller quartz-crystals of the Copper mountain porphyry of Morenci. The sericitization of the porphyry affects them but little, for the quartz-grains of the porphyries are very rarely altered by this process. In specimens taken from the oxidized zone near the surface, many of the inclusions are empty, probably due to the cracking of the grains, and the attending expulsion of the highly-compressed fluid.

The peculiar feature of these fluid-inclusions is that, as a rule, they contain, besides the gas-bubble, an extremely sharply-defined cube of transparent material, and a smaller opaque particle. The invariable recurrence of this association is a proof that these bodies are not accidental inclusions, but were present, dissolved in the fluid, at the time the quartz crystallized and imprisoned the drop of solution.

The inclusions are elliptical, irregularly rounded, or show the form of their host, that is, a hexagonal pyramid with short prism. Their size ranges from those barely visible up to 0.02 mm. in diameter; the latter being about the maximum. A frequently recurring size is 0.012 mm. Their distribution in the phenocrysts is irregular; they are not ranged along any given plane or surface. Moving bubbles are often noted in the smaller inclusions. Heating to 40° and up to 80° C. does not noticeably affect the relation of fluid cube and gas; they do not therefore consist of carbon dioxide, but of some aqueous solution. The proportion of volume between bubble and inclusion is not constant; many of them contain large gas-bubbles, while in others they may be quite small. To some extent this may be explained by leaking, but comparing a great number in very fresh rocks there certainly appears to be considerable variation in this proportion. The fluid is colorless.

As to the cube of transparent salt, it is very sharply defined and well developed. In polarized light the cube appears isotropic. Its size varies, but is usually of about the same volume as the bubble, and occupies from 4 to 15 per cent. of the volume of the inclusion. Such cubes have been sometimes ob-

served before, especially in inclusions contained in quartz crystals; and in some cases they have been proved to consist of sodium chloride. They have also been noted in quartz phenocrysts of certain Cornish granite-porphyries. In the present case the small size of the inclusions, and especially the degree of alteration and oxidation to which most of the specimens have been subjected, renders experimental determination of the salt impracticable, at least with the material in hand at present. It may be said, however, that in all probability the substance is sodium chloride. Several inclusions were measured to determine the degree of saturation when the substance was dissolved in the fluid, assuming that it is NaCl, and that the liquid is a saturated solution of the same salt at ordinary temperature. Results indicate that this was ordinarily from 5 to 20 per cent. above the maximum amount soluble in water under ordinary conditions. In one case it was found that the water must have contained 45 per cent. of salt. Most of the inclusions also contain a small opaque particle, generally measuring only a fraction of the volume of the bubble or the cube. It has no distinct form; occasionally, rod-shaped outlines may be observed, but ordinarily it is so small that it only appears as a black speck. Examined in reflected light, one unusually large particle seemed decidedly black, while another inclusion, contained also in a Copper Mountain porphyry, seemed distinctly reddish in transmitted light.

These inclusions are certainly a characteristic feature of the granite-porphyries of Morenci and Metcalf. They prove, I think, conclusively, that the acid porphyry-magma was accompanied by notable quantities of aqueous solutions containing a large quantity of a salt, which probably was NaCl; and also a smaller amount of some compound containing one or several of the heavy metals. From what follows, it is extremely probable that this compound is largely ferric oxide. It may well also contain copper, although a direct evidence of this cannot be furnished.

These observations gain in interest when it is considered that the porphyry has caused a strong metamorphism of adjoining limestone, the principal feature of which is an acquisition of silica and iron, which in all probability were given off by the porphyry. It is now shown that the magma contained heavy

metals in aqueous solution. Very likely these solutions also contained much silica, but it should be remembered that this material would most likely have been deposited, upon the cooling of the rock, on the walls of the inclusion, and in such a case it would naturally have the crystallographic orientation of the host from which the new substance could not readily be distinguished.

It is perhaps a significant fact that these inclusions are absent in the diorite-porphyries, which, as a rule, have no connection with the copper-deposits.

As to the quantitative importance of the inclusions, it may be estimated that in some specimens they make up a very perceptible percentage of the rock volume.

In Vein-Quartz.—The results of the examinations of the porphyries encouraged a search in the vein-quartz. In a greatly altered pyritic porphyry from the Butler and London tunnel at Morenci, the ground-mass is silicified and contains irregular replacement-veinlets of quartz, which were found to contain inclusions entirely similar to those in the porphyritic quartz with cubic and opaque body. In some cases three small opaque masses were found in one inclusion.

At Metcalf the granite close to the porphyry is greatly shattered and cemented by veinlets of quartz with scattered and minute foils of specularite and occasionally some pyrite. The quartz of the granite has the usual fluid-inclusions mentioned above. The cementing-veinlets of granular quartz are full of remarkably beautiful and fairly large (up to 0.02 mm.) inclusions of the usual rounded or pyramidal-prismatic form. Most of these contain bubble, cube and opaque body. The bubble varies as usual in its relative size; the cube is sharply defined and of the size described under the inclusions in porphyry. In a few of the inclusions the dark bodies are unusually large and have a rounded flat form; they are here translucent with reddish color, and there can be little hesitation in identifying them as ferric oxide or specularite. In some inclusions small grains or crystals, beside the cube, and occasionally transparent matter adhering to the wall, are also found. All this shows that the same or very similar solutions, which formed a part of the magma, deposited quartz in the immediately surrounding rock or in the solidified porphyry itself. It is clear that these solu-

tions must have been very hot and probably also under great pressure, since they held dissolved not only much larger quantities of salt than can be taken up by water at ordinary temperature, but also a substance, which probably is ferric oxide, entirely insoluble under ordinary conditions. This directly connects the solutions contained in the magma with those which deposited quartz shortly after the intrusion and shows that the general quartz-cementation which occurred, closely following the consolidation of the magma, is probably not due to atmospheric waters, but to eruptive after-effects.

In the next place, the strong pyritic, primary fissure-veins were examined, which cut across both porphyry and metamorphosed limestone at Morenci. They are associated with a little normal, coarsely crystalline, vein-quartz. Specimens from the Montezuma vein from different places showed typical vein-quartz full of fluid-inclusions, either irregularly arranged or in places following certain planes in the grains. The quartz-grains often show crystallographic outlines, are occasionally speared by smaller quartz-prisms and are associated with a few large irregular grains of pyrite. Though some of the inclusions are irregular, the majority have rounded outlines, more seldom bi-pyramidal. The size ranges up to 0.012 mm. The relation of bubble and cavity is not constant; many inclusions are empty, no doubt due to leaking, for the quartz is considerably crushed. No change is apparent upon heating the slide to 40° and even to 80° C. Almost always the inclusion contains solid bodies. Cubes, so abundant in the porphyries, are of rare occurrence and seldom well-developed. Most frequently are transparent adhesions to the wall, rod-like masses, pyramidal crystals, or irregular grains. None seem to act on polarized light, possibly on account of the minute size. In most cases the inclusions also contain one or two minute opaque bodies, which cannot be further determined. In a few inclusions the solid material is so abundant as to cause the bubble to assume an elongated form.

Entirely similar inclusions are found in the quartz of the Humboldt vein, occurring as branching-veinlets in porphyry.

The granite along Chase creek, half a mile above the foot of the Longfellow incline, contains many quartz-veins with pyrite, chalcocite and molybdenite. The quartz contains fluid-inclu-

sions, though many of the cavities are empty. Most of the inclusions contain some solid material; a few of these are imperfect cubes; most common are transparent adhesions to the walls, while many also contain opaque bodies.

These observations indicate that the quartz in the fissure-veins was formed in the presence of aqueous solutions of probably several salts. The cubes, so prominent in the porphyry and in some probably earlier veinlets of quartz, seem to be less uniformly present in those of the regular veins. It also follows that the solutions were very hot, for they contained a much larger proportion of salts than can be dissolved at ordinary temperature or even at + 100° C. The opaque bodies indicate that some combination of the heavy metals were also present in the waters.

The quartz-veinlets, connected with the processes of formation of chalcocite and of copper carbonates, contain only few and small inclusions, in which, thus far, nothing but the fluid and the bubble have been observed.

Summary of Genesis.

It has been shown that the intrusions of stocks and dikes of granite-porphyry and quartz-monzonite porphyry, which took place in late Cretaceous, or early Tertiary times, produced an important contact-metamorphism in shales and limestones of Paleozoic age, which happened to adjoin them. This metamorphism resulted in metasomatic development of garnet, epidote, diopside, and other silicates, accompanied by pyrite, magnetite, chalcopyrite, and zinc-blende. The sulphides are not later introductions, but contemporaneous with the other contact minerals.

The contact-zone received very substantial additions of iron oxides, silica, sulphur, copper, and zinc, enough to form good-sized deposits of pure magnetite and low-grade deposits of chalcopyrite and zinc-blende, all of which are entirely unknown in the sedimentary series away from the porphyry.

In view of the evidence, I consider it impossible that circulating atmospheric waters have effected these changes. The occurrences of metamorphosed rocks are manifold and found under many varying conditions; there is only one common factor and that is the presence of the porphyry. It is shown that

the porphyry magma contained much water which held dissolved various salts, among them some of the heavy metals. Sodic chloride and ferric oxide probably predominated. I believe that it contained all of the substances mentioned above, and that large quantities of this gaseous solution (for the critical temperature must have been exceeded) dissolved in the magma were suddenly released by diminution of pressure as the magma reached higher levels, and forced through the adjoining sedimentary beds; the purest and most granular limestones suffering the most far-reaching alteration and receiving the greatest additions of substance. It is thus held that a direct transfer of material from cooling magma to adjacent sediments took place. The formation of garnet indicates large gains of ferric oxide and silica. If the magmatic waters carried iron only as ferric oxide some of it must have been reduced to magnetite during the metamorphism, for the latter mineral is much more common than the specularite. These contact-metamorphic deposits often occur at the immediate contact of the main porphyry stock and the limestones. But more commonly they seem to be connected with dikes of the same porphyry close to the principal mass, these dikes being probably more highly charged with magmatic waters.

It is shown that fissures and extensive shattering developed both in porphyry and altered sediments after the congealing of the magma, and that these fissures and seams were cemented by quartz, pyrite, chalcopyrite, and zinc-blende; forming normal veins largely of the type of replacement-veins. The amount of copper contained in these is usually small, though in places possibly large enough to form pay-ore. The bulk of the veins consists of pyrite. Two classes of veins may be distinguished. The usual type is practically always connected with granite-porphyry or quartz-monzonite porphyry; it occurs in this rock or along dikes of the same. The smaller division consists of those connected in their occurrence with diabase dikes. The genesis of the former type will first be discussed.

As far as the metallic minerals are concerned there is a striking similarity between the veins connected with porphyry and the contact-metamorphic deposits, the only difference being in the magnetite, which does not occur in the veins proper and only subordinately in certain of the altered wall-rocks. A rela-

tionship is also clearly seen in the remarkable action of the vein-solutions on the adjoining wall-rock wherever this is limestone, tremolite and diopside being formed in it by replacement. On the whole, iron and silica are the main substances added, during contact-metamorphism, as well as during the vein-formation.

A study of the fluid-inclusions in the vein-quartz proves conclusively that the veins were formed by aqueous solutions and that these solutions were at a high temperature, for they contained various salts, in part, those of heavy metals, probably iron, which have separated out during the cooling of the crystallized quartz. This entirely eliminates the possibility of deposition by cold surface-waters and points to two or three eventualities. Deposition by atmospheric waters heated by contact with the cooling porphyry, or deposition by ascending magmatic waters, or, finally, by a mixture of both. In any case the metals must be derived from the porphyry, or from deep-seated sources below the porphyry, for, as stated above, the presence of porphyry is the only common factor in all occurrences. It is clear that a positive solution of these problems must be most difficult, but, here again, the fluid-inclusions offer the only direct evidence. In the quartz-seams penetrating the granite near the porphyry-contact at Metcalf, inclusions were found which are indistinguishable from those characteristic of the quartz-phenocrysts in the porphyry, and it must be concluded that the same highly heated and highly charged solutions were active in both cases. This directly connects some of the probably earlier quartz-veins with magmatic water and is evidence of considerable importance. The vein-quartz of Morenci contains inclusions which, in some cases, are identical with those in the porphyry, and in other cases slightly different from them, but always indicate highly heated solutions. The metasomatic action of the waters proves them to have been rich in silica and various salts, among them some of the heavy metals, but entirely deficient in carbon dioxide. Considering this evidence, together with the similarity of the products of these processes with those of contact-metamorphism, I think it certain that parts of the mineral solutions were directly derived from and formed part of the porphyry magma, and I believe it is probable that they were entirely derived from this source.

It seems likely, that the fissuring, which took place after the cooling, opened vents of escape for magmatic waters under heavy pressure at lower levels, and that they ascended on these fissures depositing the heavy metals and the silica and acquiring at the same time carbon dioxide from the sediments which they traversed.

As to the depth at which deposition took place, no positive evidence is available on account of lack of data concerning the extent of erosion. But from stratigraphic consideration, it is not likely that the depth from surface was more than two or three thousand feet. The cause of the deposition was no doubt a decrease in temperature, just as the deposits are formed in the cooled fluid-inclusions. I think it likely that, in most cases, the solutions were present as liquids, for, assuming that the waters communicated with the surface, neither pressure nor temperature could have been high enough to reach the critical point. Probably this does not matter very much, for the properties of water appear to be very similar for some distance each side of this point. Copper, iron and zinc are practically the only important metals present in the main deposits close to the main porphyry stock; but it is interesting and suggestive to note that gold begins to appear in veins which are located some distance away from the central porphyry.

The veins connected with diabase dikes are few in number, and the opportunity for their study has been limited. It seems risky, therefore, to express a definite opinion on their genesis, except that the copper and iron sulphides in all probability were derived from the diabase itself, either by means of magmatic or heated atmospheric waters.

The deposits thus far described are, in general, of low-grade, only rarely containing enough copper to be classed as economically important. Those in shale or limestone consist of disseminated sulphides, in places irregularly concentrated, or accumulated along certain strata, according to the well-defined tendency of contact-metamorphism. Those in porphyry consist of heavy veins of pyrite and a small amount of other sulphides, surrounded by zones of dissemination of the same sulphides.

It remained for the surface-waters, as erosion gradually exposed the deposits, to alter and enrich them in manifold forms.

From the evidence presented above, it must be concluded

that some of the deposits, especially the fissure-veins, were laid bare by erosion, and attacked by surface-waters at an early date, probably before the principal faulting-movement, and certainly before the eruption of the Tertiary basalts and rhyolites. Oxidation has thus acted on them for a very long period.

The irregular and tabular deposits of oxidized ore in limestone and shale have been formed, partly by direct oxidation of sulphides and partly by the influence of sulphate solutions derived from widely disseminated chalcopyrite due to contact-metamorphism; a great enrichment has taken place, due to decrease of volume and addition of copper from the circulating sulphate solutions. Some of the oxidized deposits in shale, however, may be wholly due to adsorption, exerted by the kaolin in the shale on these sulphate solutions.

In the veins, and especially in those which traverse the porphyry stock, or follow porphyry dikes, the history is more complicated. It has been shown that oxidation dates back to Tertiary times, and that the water-level then was considerably higher than it is at the present time. By action of descending sulphate solutions on pyrite, chalcocite was deposited very extensively, and very likely the great vertical extent of the chalcocite-zone, ordinarily from 200 to 500 ft., is due to slow and gradual changes in the water-level. Disintegration and erosion removed the iron-cap (the products of direct oxidation of the primary vein) and began to oxidize the exposed chalcocite-zone. In practically all of the veins, the surface-zone of poor ore is due to the direct oxidation of chalcocite. The solutions from this part descend and add richness to the upper part of the remaining chalcocite-zone. But at the present low stand of the water-level, and the exceedingly dry climate, the lower limit of the chalcocite-zone is probably almost stationary.

The copper-deposits of Clifton and Morenci are thus believed to have been formed primarily by mineral-laden magmatic waters, partly acting as gas and partly as liquids, and in both cases derived from a magma of granite-porphyry. These solutions were evidently directly released from the magma without a preliminary concentration in pegmatitic or aplitic dikes, which, indeed, do not occur in this district in association with the porphyry. It is perhaps superfluous to emphasize

that these conclusions are not generalizations, and that this mode of origin is not necessarily that of all other metalliferous veins.

The earlier processes of magmatic origin produced low-grade deposits of pyritic ores, and the final concentration to payable ore-bodies has chiefly been effected by descending and oxidizing surface-waters of atmospheric origin.

Genetic Classification.

The following scheme of genetic classification of the deposits is presented:

I. First Epoch. Formed during the consolidation of porphyry by laterally moving water-gas.
   a. *Contact metamorphic deposits.*—Irregular or tabular disseminations in shale or limestone, sometimes following stratification planes or dike contacts. Ores consist of pyrite, chalcopyrite, zinc-blende and magnetite. Generally unpayable.

II. Second Epoch. Formed shortly after the consolidation of the porphyry by ascending hot, magmatic waters. Continued on porphyry, granite or more or less altered sedimentary rocks.
   a. *Fissure-veins.*—Lode-systems or single veins. Central seams of pyrite, chalcopyrite and zinc-blende accompanied by wide zones of sericitization and pyritization of the porphyry. Generally unpayable.
   b. *Stock-werks and irregular disseminations.*—Some character of mineraliztion. Unpayable.

III. Third Epoch. Fissure-veins formed by ascending-waters shortly after the intrusion of diabase.

IV. Fourth Epoch. Deposits formed by descending atmospheric waters acting on Classes I., II. and III.
   a. *Concentrations by direct oxidation and secondary chalcocite-deposition in type I.* Payable.
   b. *Concentrations by direct oxidation and secondary chalcocite-deposition in type IIa.* Payable.
   c. *Concentrations by direct oxidation and secondary chalcocite-deposition in type IIb.* Payable.
   d. *Deposits caused by sulphate-waters along otherwise barren fault-planes.* Occasionally payable.
   e. *Deposits caused by sulphate-waters along shale beds.* Partly payable.

Evidences of Plication in the Rocks of Cananea, Sonora.

BY WILLIAM P. BLAKE, TUCSON, ARIZONA.

(Lake Superior Meeting, September, 1904.)

THE copper-bearing rock formations of Cananea, Sonora, Mexico, present conclusive evidences of extensive and sharp plication. Some of these evidences are here briefly stated.

1. There is a succession of approximately parallel outcrops of an altered limestone, alternating with dikes and masses of porphyry. The great copper-ore deposits are in close association with these ridges of limestone, which trend northerly and southerly and stand nearly vertical. This parallel succession is so well-defined that the management of the mines has, for convenience, recognized and named five zones, known from the chief mines, in their succession from east to west, as the "Cobre-Grande," "Veta-Grande," "Esperanza," "Capote," and "Puertocitas" belts. The intervening and separating formations are coarsely crystalline feldspar porphyries, highly basic, of light color, which may be regarded as intrusive and, in part at least, as the cause of the metamorphosed condition of the limestone.

2. These cupriferous limestone-zones are similar in constitution, are similarly charged with copper-ore (in association with quartz as a gangue, or veinstone), and are marked by similar croppings of gossan.

3. Distinctly folded outcrops of the surrounding strata can be observed, especially on the eastern border of the copper-bearing district north of the Democrita ground, and not far from the Veta-Grande.

4. The limestone ridge adjoining the Veta-Grande ends a short distance north of the mine, and appears to be the extreme end of a synclinal trough.

5. The Capote limestone, the most western of the croppings of the central copper-bearing area, is contiguous to stratified sandstones and quartzites, uplifted at high angles, but showing an easterly dip. The Capote limestone is apparently conformable

with these stratified beds, and a member of the series. The limestone terminates abruptly at the north, towards the Indianapolis ground, in a form suggestive of a vertical faulting-plane.

6. Heavy croppings of gossan, covering a considerable area on the Indianapolis ground, have a general triangular form with the apex westward towards a ridge of limestone. A prolongation of this limestone bounds the hills of gossan on all sides except the east, where both the limestone and the gossan terminate abruptly in a vertical fault. The conditions clearly indicate that, in this Indianapolis area of massive iron-ore gossan, we have the western end of a synclinal trough. The relations of this fold and of the fault-plane to the contiguous Capote ground require more extended study. There is reason to believe that the formations were originally continuous and united, but have been separated by extensive faulting-movements.

7. The formations of the area beyond the Capote and the Indianapolis, westward to the Puertocitas, are very different from those of the area which are partly described above. There is here a higher region of distinctly folded limestone, the original stratification of which has not been obliterated by metamorphic agencies. It is a copper-bearing formation, in which the ores are inter-bedded. At the Puertocitas, several miles beyond the Indianapolis and the Capote, the copper-ore is largely associated with garnet.

Leaving out of view the Puertocitas, and other deposits west of the Capote and the Indianapolis, the central copper-bearing region of Cananea may be regarded as a depressed basin, occupied by remnants of a limestone, once much more widely extended, a large part of which has been removed by erosion, together with the copper-ore associated with the quartz vein-stone. The abundance of quartz accompanying the copper-ore gives to these deposits the nature of true lodes, which apparently are most highly developed where the plication has been sharpest and deepest.

Improvements in the Mechanical Charging of the Modern Blast-Furnace.

BY DAVID BAKER, PHILADELPHIA, PA.

(Lake Superior Meeting, September, 1904.)

OUR large modern blast-furnaces, equipped with ore-bins, larries and mechanical means for putting stock into storage, withdrawing it therefrom, and charging it at the tunnel-head, are indeed wonderful in capacity; yet, it is admitted that, with some notable exceptions, they have disappointed us in economy—that is, in the items of low fuel-consumption, uniformity of product, freedom from slips, and reduced cost of repairs and relining.

The manager who estimated the cost of pig-iron to be made in a modern plant, by taking the costs of the hand-filled furnace and deducting the amount to be saved by the mechanical appliances, has been surprised, in many cases, to find that unfavorable results in these items of economy have greatly reduced his expected gains.

Is there any reason why the fuel-consumption should be greater for a modern furnace than for an older stack using the same mixture? Certainly the equipment is much more complete, and the trouble, therefore, must lie in the management or in the character of that equipment. Everybody knows that intelligent and vigilant management is a prime requisite; that any neglect of a detail, though it may not be felt that day or week, will surely sooner or later make its appearance as an increased item of cost. But to hold the furnace-manager responsible for the relatively poor work of the modern blast-furnace would be, I am convinced, a great injustice. For its management has received much more study and care than was given to the older types. Many furnace-managers have lost their health through worrying over this problem, and some have lost their positions, being held responsible for results made inevitable, as I believe, by the equipment provided for them.

We should therefore investigate the character of this equipment.

The modern stack tends to work irregularly; slips are too frequent and too much "off"-iron is made; the furnace "swings" too often and too much. I think it is this irregular working that causes the greater fuel-consumption; and according to experience, when a furnace begins to work irregularly, one of the first things to be examined in searching for the cause was the distribution of the stock on top.

When we were filling furnaces by hand, how often was irregular furnace-work traced directly to dishonest top-filling! One of the great objects of mechanical filling was not only the saving of labor, but also the elimination of irregular charging, which was known to exist where the work was done by hand.

For many years before closed-top furnaces became the rule, we accomplished with the simple bell-and-hopper charger, when honestly handled, a very even distribution of the charge. Each barrow-load of material was dumped in a certain space on the bell, to be changed at the next charge. It was a common practice to charge the coke, stone and ore in layers. But to do so, we depended on the honesty of the top-chargers; and too frequently, in order to save distance in wheeling, the larger part of the stock was dumped on the side nearest the hoist, or, to avoid a place where hot gas was escaping from a leak in the bell or hopper, preference was given to the cooler side.

In seeking for a mechanical device for furnace-charging, no attempt was made to dispense with the ordinary bell-and-hopper top, but various plans were proposed to utilize the skip-hoist, on account of its simplicity and great capacity, by adding some receiving-hopper, that should discharge its contents evenly on the main bell-and-hopper.

The problem is to dump the skip-bucket, containing material of different sizes, at a given point on top of the furnace, so that the lumps and fines will be equally distributed around the furnace-bell. To accomplish this, sundry contrivances have been constructed, consisting of a receiving-hopper into which the skip or skips dump, and which discharges into a circular receptacle (a cylinder or frustrum of a cone), closed at the bottom by a small bell, the whole apparatus being placed immediately above the

main bell of the furnace. Each different design was supposed to have its special merits, but, in most cases, was simply the result of an effort to avoid somebody's patent.

Another improvement in top-charging is the use of a gas-seal to prevent the escape of gas when the charge is lowered into the furnace. This is, in fact, not only an economy, but also a necessity, in order to protect the superstructure on the furnace.

Skip-Hoist Experiments.

In 1890, before the form of top for furnaces No. 9 and No. 10, then under construction at the South Works of the Illinois Steel Co., had been determined, I made a half-size model of a furnace-top of the double-bell type, arranged for experiments with the single or double skip, under conditions approximating as closely as possible those of full size.

Fig. 1 shows the model in elevation and Fig. 2 a plan of the hopper, scale 0.5 in. = 1 ft.

The hopper, instead of being circular in section, was made up of 12 plane sides and divided into 12 equal spaces by partitions made of thin wood.

In using this model, no experiments were tried with coke, or stone. A study of the distribution of the ore, it was thought, would be a guide as to the distribution of the limestone and coke. The ore-charge used was a proportional part of the charge then being filled into Furnace No. 5 of the Illinois Steel Co. corresponding to the area of the hopper of the model, when compared with a furnace of twice the diameter, and consisted of:

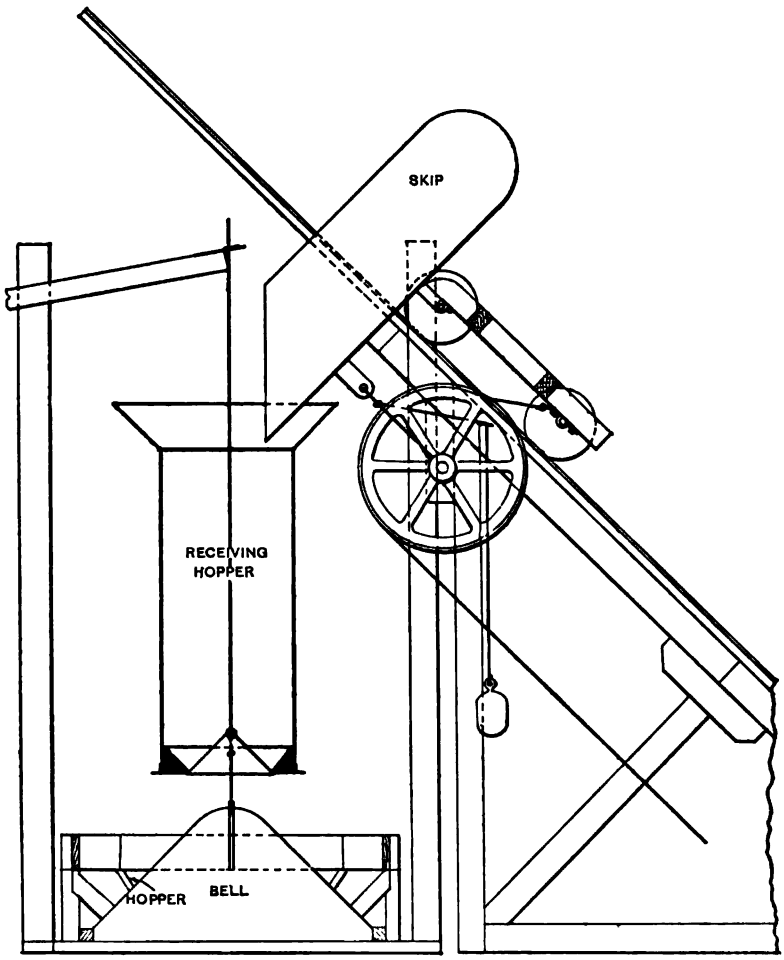
Ore.	Pounds.
Fayal,	1,937
Lake,	655
Pewabic-Genoa,	417
Champion,	715
Chapin,	327
Blue-billy,	149
Total,	4,200

The Champion lump-ore, and the Pewabic-Genoa ore, were broken down to half-size.

Single-Skip Trials.

Three experiments were made with the single-skip, A, B, C, in which the ore was charged in the following order: Blue-billy at the bottom of the skip-bucket; then Chapin, Cham-

FIG. 1.

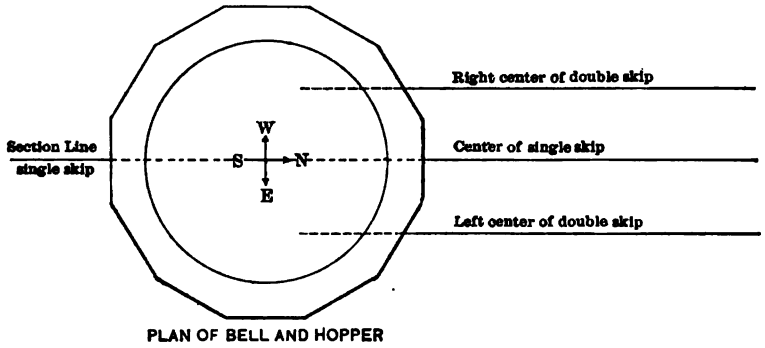


HALF-SIZE MODEL OF CHARGING APPARATUS.

pion, Pewabic-Genoa, Lake, and Fayal ore. After dumping the skip, the ore occupied generally, in the cylinder, of course, the reverse of its previous order in the skip; but the blue-billy was on the north side, next to the skip.

In dumping this charge from the cylinder into the main hopper, the blue-billy was found to be all in the spaces 1, 2, 3, 11 and 12—fully 40 per cent. of it being in division 1, and the rest in gradually decreasing amount from the center of No. 1

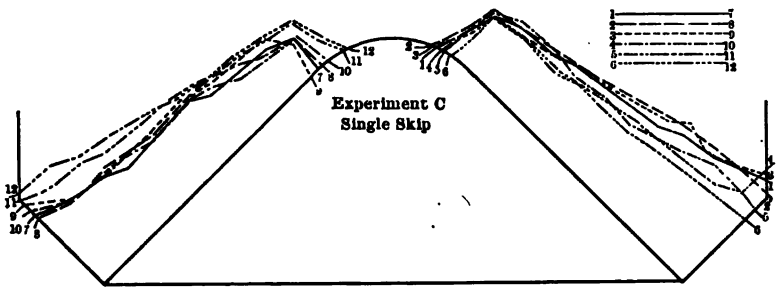
FIG. 2.



HALF-SIZE MODEL OF CHARGING APPARATUS.

to the right and left (see Figs. 3, 4 and 5, of which the last shows also the profile of the ore in the cylinder when a single skip is used). In the main hopper, the ore was apparently

FIG. 3.



PROFILE OF ORE IN HOPPER, CHARGED BY A SINGLE SKIP.

heavier on the side next to the skip, and lighter on the opposite side. The hard ore seemed about evenly distributed.

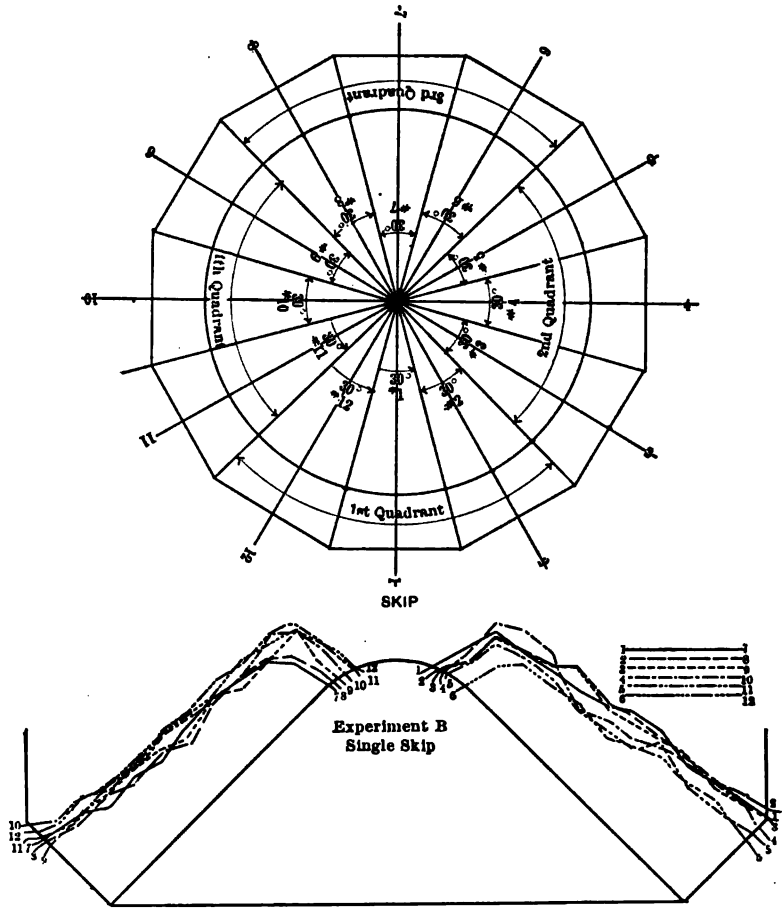
Double-Skip Trials.

For these it was necessary to increase the width of the hopper which received the discharge from the two skips. Three

experiments, A, B and C, were tried, in which the skip-car was filled as follows:

Experiment A.		Experiment B.		Experiment C.
Left-load.	Right-load.	Left-load.	Right-load.	
.....	Blue-billy.	Blue-billy.	.....	Same as A.
.....	Chapin.	Chapin.	.....	
Fayal.	Champion.	Champion.	Fayal.	
.....	Pewabic-Genoa.	Pewabic-Genoa.	.....	
.....	Lake.	Lake.	.....	

FIG. 4.



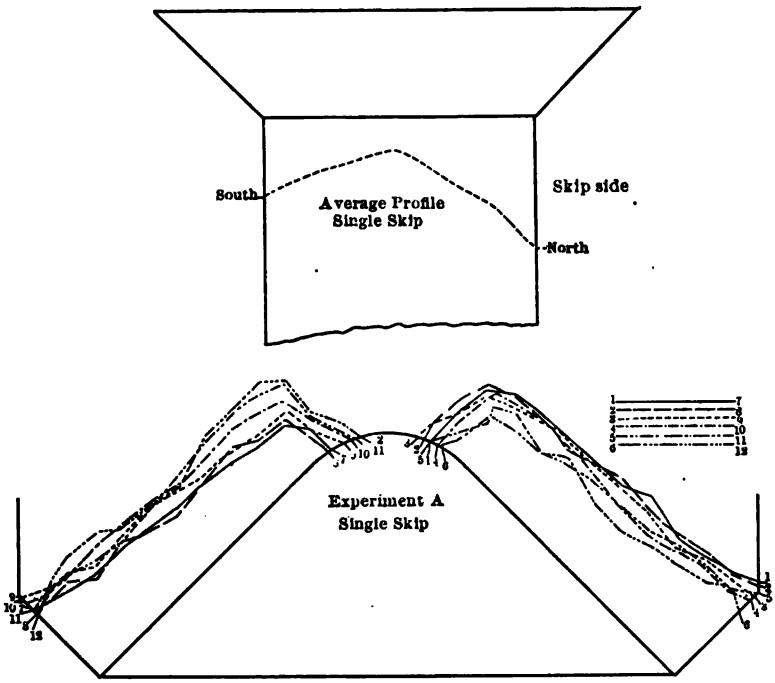
PLAN OF DIVISIONS IN HOPPER AND PROFILE OF ORE IN HOPPER, CHARGED BY A SINGLE SKIP.

Fig. 6 shows the profile of the ore in the cylinder in experiments A and B. Figs. 7, 8 and 9 show the profile of the ore in the main-hopper trials A, B, C. In these experiments the

blue-billy seemed to be mixed up with the other ore, and its distribution could not be clearly traced.

It was noticed, however, that the smallest deposit of ore in the main hopper corresponded to the smallest deposit in the cylinder. This is the reverse of the condition shown in the trial with the single skip. Moreover, there was with the single skip more complete separation of the lumps and fines. Some trouble was experienced in all the experiments from the

FIG. 5.



PROFILE OF ORE IN CYLINDER AND IN HOPPER, CHARGED BY A SINGLE SKIP.

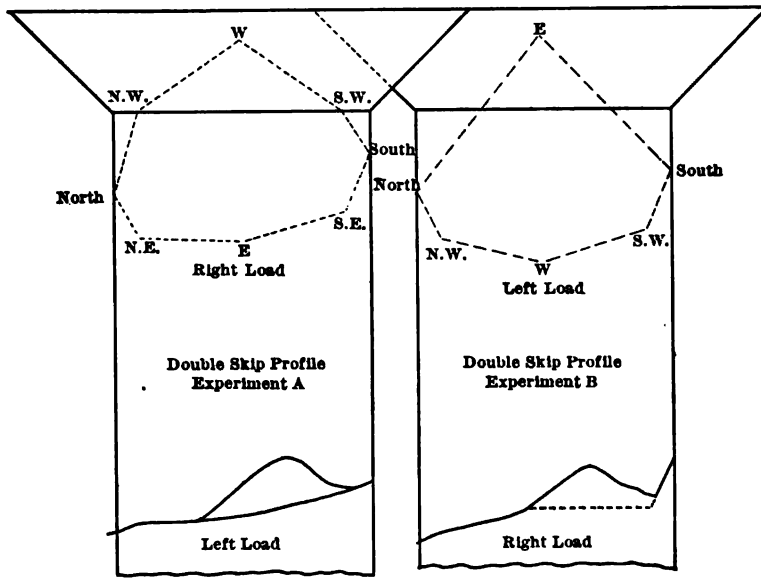
"bridging" of the ore in the cylinder; but in practice no such effect has been noticed with a cylinder of twice the diameter.

In order to ascertain the distribution of the lumps in the main hopper, the ore in each compartment was screened through a 1-in. mesh sieve.

Table I. shows the distribution of the ore in weight. The compartments are numbered, beginning on the side next to the skip.

Table II. shows the weights figured by quadrants, and Table III. shows these figures expressed in percentage, while Table IV. shows the variation above and below the average in each case.

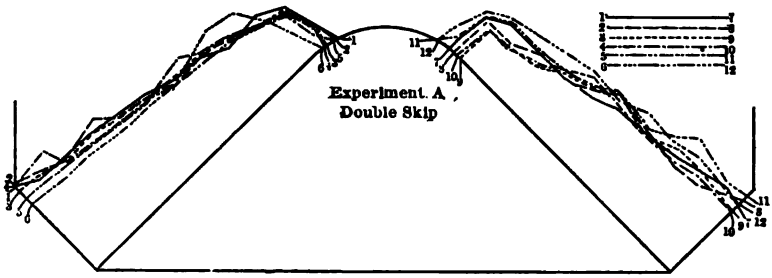
FIG. 6.



PROFILE OF ORE IN CYLINDER IN EXPERIMENTS A AND B.

Taking quadrants 1 and 3, that is, the ore under the skip and that opposite, in the column showing the difference in the

FIG. 7.

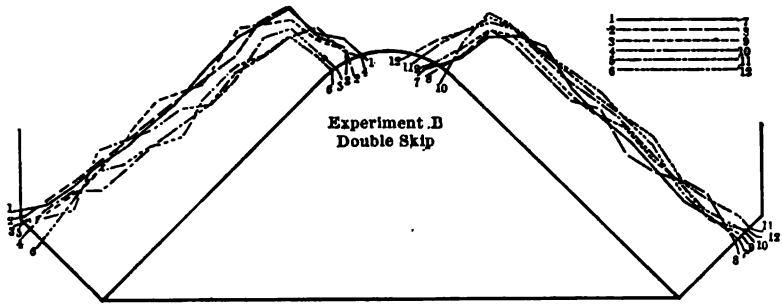


PROFILE OF ORE IN HOPPER, CHARGED BY A DOUBLE SKIP.

weight of total ore in Table IV., when using the single skip, we find 261 lb. difference in weight, or 6.12 per cent., while between the other two quadrants there is only 17 lb. difference,

or 0.42 per cent. In the case of the double-skip quadrants Nos. 1 and 3 show 208 lb. and 2 and 4 show 78 lb.; or 5.09 per cent. difference for 1 and 3, and 1.85 per cent. for 2 and 4, but when we take the rights and lefts together, with the double skip, the difference shows as 18 lb. between 1 and 3 and 86 lb. between

FIG. 8.

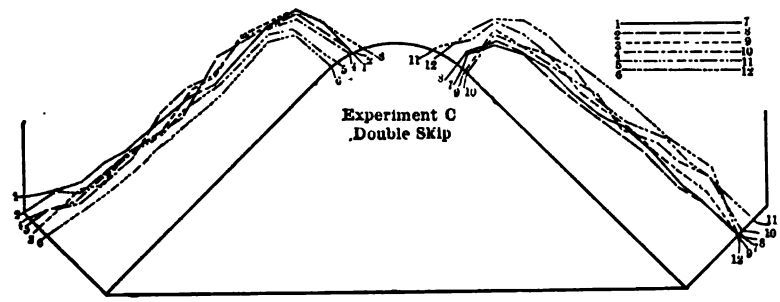


PROFILE OF ORE IN HOPPER, CHARGED BY A DOUBLE SKIP.

2 and 4, or 0.42 per cent. between 1 and 3 and 2.10 per cent. between 2 and 4, so a great advantage is obtained by changing the hoisting of the charge from left to right and vice versa.

Now, if we look at the lump columns, we see a difference in case of single skip of 121 lb. between quadrants 1 and 3 and

FIG. 9.



PROFILE OF ORE IN HOPPER, CHARGED BY A DOUBLE SKIP.

4 lb. between 2 and 4. In case of double skip, 42 lb. between 1 and 3, and 18 lb. between 2 and 4, showing in rights and lefts 28 lb. difference between 1 and 3 and 54 lb. between 2 and 4. This shows that the double skip makes a better distribution than the single, and that the distribution of the double skip,

TABLE I.—*Average Ore-Weights, Pounds.*

30° Divisions of Hopper.	Ore Held by 1" Mesh. Screen.			Ore Passing 1" Mesh. Screen.			Total Ore.		
	Skip.			Skip.			Skip.		
	Single.	Double.	R. and L. Double.	Single.	Double.	R. and L. Double.	Single.	Double.	R. and L. Double.
1	143	133	126	237	279	269	380	412	395
2	155	122	115	240	250	251	395	372	372
3	141	124	120	252	229	240	393	353	360
4	144	150	145	237	265	262	381	415	407
5	117	114	110	199	202	196	316	316	306
6	104	114	127	173	197	211	277	311	338
7	96	104	103	184	207	209	280	311	312
8	94	100	104	201	173	174	295	273	278
9	118	110	111	199	190	184	317	300	295
10	141	141	152	234	220	218	375	361	370
11	140	120	126	241	227	222	381	347	348
12	117	106	106	212	214	218	329	320	324
All	1,510	1,438	1,445	2,609	2,653	2,660	4,119	4,091	4,105
Average	126	120	120	217	221	222	343	341	342

TABLE II.—*Table I. Ore-Weights Consolidated to Quadrants.*

This Consolidation Reduces Errors of Division = Errors in Weights, 50 Per Cent.

Quadrants.	Ore Held by 1" Mesh. Screen.			Ore Passing 1" Mesh. Screen.			Total Ore.		
	Skip.			Skip.			Skip.		
	Single.	Double.	R. and L. Double.	Single.	Double.	R. and L. Double.	Single.	Double.	R. and L. Double.
First	415	361	346	689	743	744	1,104	1,104	1,090
Second	402	388	375	688	696	698	1,090	1,084	1,073
Third	294	318	335	558	577	593	852	895	928
Fourth	399	371	389	674	637	625	1,073	1,008	1,014
All	1,510	1,438	1,445	2,609	2,653	2,660	4,119	4,091	4,105
Average	378	359	361	652	663	665	1,030	1,023	1,026
Above by Percentages.									
First	10.07	8.82	8.43	16.74	18.16	18.12	26.81	26.98	26.55
Second	9.76	9.48	9.13	16.70	17.01	17.00	26.46	26.49	26.13
Third	7.15	7.78	8.16	13.54	14.11	14.45	20.69	21.89	22.61
Fourth	9.68	9.07	9.48	16.36	15.57	15.23	26.04	24.64	24.71
Total	36.66	35.15	35.20	63.34	64.85	64.80	100.00	100.00	100.00
Average	9.16	8.79	8.80	15.84	16.21	16.20	25.00	25.00	25 00

TABLE III.—*Comparison by Percentages of the Ore Held by and Passing, the 1-In. Mesh Screen.*

Quadrants.	Total Ore Held by 1" Mesh Screen = 1.			Total Ore Passing 1" Mesh Screen = 1.		
	Skip.			Skip.		
	Single.	Double.	R. and L. Double.	Single.	Double.	R. and L. Double.
First	27.48	25.11	23.94	26.41	28.01	28.00
Second	26.62	26.98	25.95	26.37	26.23	26.24
Third	19.47	22.11	23.19	21.39	21.75	22.29
Fourth	26.43	25.80	26.92	25.83	24.01	23.47
Total	100.00	100.00	100.00	100.00	100.00	100.00
Average	25.00	25.00	25.00	25.00	25.00	25.00

Variables.	Single Skip Experiments.			Double Skip Experiments.		
	A.	B.	C.	A.	B.	C.
Ore	Frozen.	Charged and released, wet. Divided and weighed, frozen.	Damp.	Frozen.	Damp.	Wet.
Weather.	Freezing.	C. and F., rainy. D. and W., freezing.	45° F., dry.	Freezing.	40° F., mild.	45° F., showers.

while better, is not correct when changing the hoisting of the ore from right to left. Inasmuch as the speed of hoisting with the model was 20 ft. per min., a more rapid rate should prevent in a great measure the separation of the lumps and fines.

CONCLUSION.

From these experiments it is evident that the double skip, with the form of top described, gives a more even distribution of the lumps and fines than the single skip. As a matter of fact, the double form has been most widely adopted.

As a result of our experiments, we therefore considered the distribution with the double bell, when filled by the double skip, to be good enough for our practice, counting upon the more rapid dumping of the actual furnace-skip to prevent sorting of

TABLE IV.—*Variation from Evenly Distributed. + Above. — Below. By Weight.*

Quadrants.	Table II.									Table III.					
	Ore Held by 1" Mesh Screen.			Ore Passing 1" Mesh Screen.			Total Ore.			Ore Held by 1" Mesh Screen.			Ore Passing 1" Mesh Screen.		
	Skip.			Skip.			Skip.			Skip.			Skip.		
	Single.	Double.	R. and L. Double.	Single.	Double.	R. and L. Double.	Single.	Double.	R. and L. Double.	Single.	Double.	R. and L. Double.	Single.	Double.	R. and L. Double.
First	+ 37	+ 1	— 15	+ 37	+ 80	+ 79	+ 74	+ 81	+ 64	+ 37	+ 2	— 15	+ 37	+ 79	+ 79
Second	+ 25	+ 29	— 26	+ 35	+ 33	— 72	+ 60	+ 62	— 98	+ 24	+ 29	— 26	+ 36	+ 33	— 72
Third	— 84	— 41	+ 13	— 94	— 86	+ 33	— 187	— 127	+ 46	— 83	— 41	+ 13	— 94	— 86	+ 33
Fourth	+ 21	+ 11	+ 28	+ 22	— 27	— 40	+ 43	— 16	— 12	+ 22	+ 11	+ 28	+ 21	— 27	— 40
Above by Percentages.															
First	+ .91	+ .03	— .37	+ .90	+ 1.95	+ 1.92	+ 1.81	+ 1.98	+ 1.55	+ 2.48	+ .11	— 1.06	+ 1.41	+ 3.01	+ 3.00
Second	+ .10	+ .70	— .64	+ .86	+ .80	— 1.75	+ 1.46	+ 1.50	— 2.39	+ 1.62	+ 1.98	— 1.81	+ 1.37	+ 1.23	— 2.71
Third	— 2.03	— 1.01	+ .33	— 2.28	— 2.10	+ .80	— 4.31	— 3.11	+ 1.13	— 5.53	— 2.89	+ .95	— 3.61	— 3.25	+ 1.24
Fourth	+ .52	+ .28	+ .68	+ .52	— .65	— .97	+ 1.04	— .37	— .29	+ 1.43	+ .80	+ 1.92	+ .83	— .99	— 1.53
Variation Between Opposite Quadrants by Weight. (Pounds.)															
First and third	121	42	28	131	166	46	261	208	18	120	43	28	131	166	46
Second and fourth	4	18	54	13	60	32	17	78	86	2	18	54	15	60	32
Variations Between Opposite Quadrants by Percentages.															
First and third	2.92	1.04	0.70	3.20	4.05	1.12	6.12	5.09	0.42	8.01	3.00	2.01	5.02	6.26	1.76
Second and fourth	0.08	0.41	1.32	0.34	1.44	0.78	0.42	1.85	2.10	0.19	1.18	3.73	0.54	2.22	1.18

the material, as well as to make up for defects of distribution developed during experiments with the model.

As soon, however, as the furnaces provided with mechanical charges were started, it became apparent that most of the lumps were being thrown to the side opposite the skip; and, though deflectors were promptly placed in the receiving-hoppers, the results have been unsatisfactory, and the life of the linings (which cut out on the side opposite to the skip) has been short; this is the history of nearly all double-bell chargers. Some of them cause the furnaces to cut out faster than others, by reason of local conditions, such as the physical condition of the coke, limestone and ore; but with practically all of them, the lining tends to wear faster on one side—in the majority of instances, the side opposite to the skip. It may be replied that this is not true of the Duquesne furnaces, Pa.; and I must admit that very creditable work has been done there with the double bell. The reason seems to me apparent. The skip at these furnaces does not dump, but discharges from the bottom over a small bell. This is a potent preventative of "sorting;" the most important fact is that the same skip-bucket is filled in the stock-house from two opposite sides. If it were filled by shutes on one side, the sorted lumps would roll to one side and eventually land in the furnace on that side. If the skip-bucket at Duquesne could be filled from four equi-distant points of a circle, the distribution would be still better. A still simpler device might perhaps secure the desired purpose.

Destruction of Linings.

Furnace managers have been perplexed as to the cause of the rapid destruction of linings in modern blast-furnaces. By some, the blame has been placed on the quality of the fire-brick used. They say we are not getting as good fire-brick as when there was more competition among brick-makers. Whatever foundation there may be for this statement, it is surely not possible for all the worst bricks to be laid on one side of a furnace; and there has been no complaint about the durability of the bricks used in the hearth and bosh; therefore I believe we must look elsewhere for the source of the trouble.

An effort has been made to overcome the excessive wear of the lining by placing cooling-plates in the lower part of the in-

wall, the excessive wear being often first manifested at a point 15 or 20 ft. above the top of the bosh. I do not, however, believe that this excessive wear can best be prevented in that way. To my mind, the hot working on one side of the furnace indicates an interior condition calling for more than an external remedy.

In my experience I have found that wherever the filling gave a column of lumps on one side of the furnace, at that point the wall would cut away. Some years ago the blast-furnaces of the Maryland Steel Co. were filled by a car-system which dumped a ton of ore in a conical pile at four equi-distant points on the hopper.

When the first furnace filled by this method was blown out, the in-wall was found to have been cut into rectangular shape; the corners of the rectangle being directly under the equi-distant points between the piles of ore in the hopper. It was at these points that the lumps rolled, forming four chimneys of coarse stock in the furnace, through which the gas channeled. It is needless to add that the furnace-work of this stack, before blowing out, was decidedly unsatisfactory.

My experiments at Sydney, given in my recent paper on Stock Distribution and Its Relation to the Life of a Blast-Furnace Lining,[1] show that this cutting may be remedied by changing the distribution on top.

Now, since these troubles are not the rule with hand-filled furnaces, they must be due to the distribution of the stock in the mechanically-filled furnaces.

Many managers argue, however, that the furnace worked well enough until the shell got hot. To this, I would reply that the bad effect of unequal stock-distribution is cumulative, and becomes more evident with growing irregularity in the wall. What may first appear as increased fuel-consumption only, develops later into casts of "off" iron.

If bad stock-distribution may cause irregular wear of the furnace-lining, what have we lost in changing from hand-filled to machine-filled furnaces? Is it possible to distribute the stock evenly, both as to quality and quantity, by mechanical means; and if so, how? I have seen mechanically-filled fur-

[1] See p. 244.

naces in which, so far as distribution of quantities was concerned, nothing better could be desired, and yet the cutting action on the in-wall was excessive. The quantity being equally divided, the only other thing to look for is the mechanical quality. Are the lumps and fines equally divided in the distribution? In hand-filling there is a rotation of the stock, buggies being moved around to new points at each charge. It is this rotation which I regard as vital, and as having given the satisfactory results of hand-filling.

In watching, on the top of the ordinary skip-filled furnace, the distribution of the stock with double-bell charger, it will be noted that dry material, moving with a higher velocity than wet, is thrown farthest in dumping; and, moreover, that lumps have also a greater velocity than fines. This applies also to the coke, in which braize is always present, especially after the repeated handlings to which coke is subjected in the modern filling-device. As the skip goes up the incline, all this dirt is shaken down, and follows in dumping the bottom-contents of the skip, therefore being deposited immediately below the dumping-point. In other words, we have in the dumping-skip and receiving-hopper of the usual double-bell charger, a very efficient sorter which deposits the fines in one place and the lumps in another.

To correct this, many devices, such as adjustable shutes, deflectors, etc., have been tried without completely removing (though some may have mitigated) the difficulty. One of the most common is the long cylinder placed immediately below the receiving-hopper. The smaller the diameter of this cylinder the more effective will be its aid; but there are practical limits to the reduction in diameter of this cylinder, and hence it never fully secures a perfect proper distribution of stock.

In the majority of cases, the larger part of the lumps reach the side of the furnace opposite to the skip, causing the gas to channel in that direction, and raising the melting-point on that side; since the other side, not having received so much gas-treatment, is not so soon ready for the hearth. Hence the furnace works cooler on that side, and tends to "build up" on the walls, and to "scaffold" there, by reason of the larger quantity of fine ore and coke there present.

Unquestionably, one of the most potent causes of slips and irregular product is the gas-channeling, caused by irregular distribution of the fines in the furnace; and the consequent presence, inside the stack, of two separate smelting-operations —the process on the side having the most lumps progressing faster than the one on the other side, where there are more fines. I have heard metallurgists claim that the slips and irregularities under consideration are due to the great volume of blast in the modern stack; but I have always found that the more uniform results are to be obtained by keeping the furnace busy, and I always looked for irregularity when obliged to reduce materially the quantity of the blast.

But if, in the same furnace, two separate smelting operations are going on (one side being hotter than the other), increase of the blast beyond a certain volume will simply bring the cold side down too fast; the furnace will scour on one side; iron oxide will go into the cinder; and some accumulations on that side may fall. As a result, the furnace may become cold, and some tuyeres may be lost.

There is nothing to determine the course of the gas-currents, as they pass upward from the melting-zone in the furnace, except the lines of least resistance; and, if most of these are on one side of the furnace, that side will get most of the gas and will consequently "work hotter." It can readily be seen that a slight difference in resistance, if constantly in one direction, will result in diverting the gas-currents, thus causing irregular furnace-work, slips, scaffolds and cutting of the in-wall on the hot side. In my judgment, the only radical cure of this trouble is the different distribution of successive charges, preventing the segregation of the lumps on any one side of the furnace.

From a faulty distribution comes a great train of evils: higher fuel-consumption; irregular work; slips; and rapid destruction of the furnace-lining. Correct the distribution and diminish the breakage of coke, and the modern stack will do better work in every particular than the older hand-filled furnace, and will yield a fair return for the money invested in labor-saving appliances.

After long investigation of the mechanical filling of blast-furnaces, I am satisfied that it is impossible to obtain a satis-

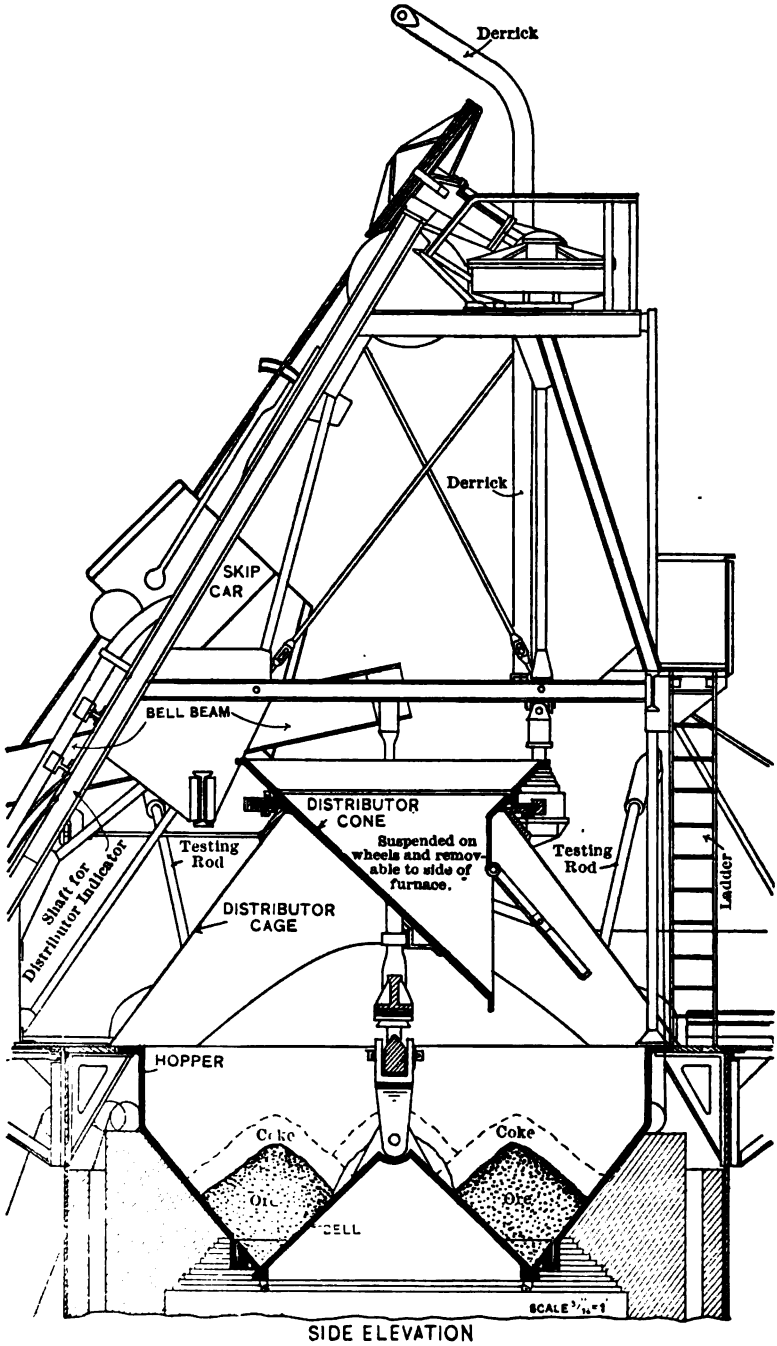
factory distribution of the stock with the double-bell charger and dumping-skip, however varied with deflectors, adjustable chutes, reducing-hoppers, or any other mechanism depending for adjustment, etc., upon an operator.

After my paper on my experience at Sydney was written, the furnace which had the latest improvements in top-charging gave signs of the burning-out of the in-wall on one side, showing that these improvements, though helpful, had not completely remedied that trouble.

In searching for some means of duplicating hand-filling mechanically, I concluded that, since the skip could not readily be rotated about the furnace-top, like the barrow in hand-filling, we must try to rotate the furnace-top so as to produce the same effect—in other words, to obtain the advantages of hand-filling and avoid its disadvantages, we must make an automatic arrangement, working in unison with the skip-hoist, and not dependent upon an operator.

The top invented some years ago by Alex. E. Brown, of Cleveland, answers all these conditions. Here, a revolving-hopper is operated by the skip-hoist rope-wheel. This hopper, provided with a wide chute, is moved into position while the empty skip is descending; a ratchet-arrangement in the drive preventing any movement in an opposite direction. This device, which may be used with single or double skip, directs each skip-load into a different section of the hopper. For instance, the coke-charge may be sent up in four skips and deposited in four equidistant parts of the hopper, thus making, owing to the wide discharge-nozzle, a complete ring of coke in the hopper. After the coke has been lowered into the furnace, the ore and limestone are sent up in the next four skips, thus making a complete ring of ore and limestone in the hopper. Now to make this arrangement even better than hand-filling, this hopper, instead of moving just 90 deg. between skip-loads, is moved, say, 4 deg. more, so that the material of one charge is dumped in a slightly different place from the same material in the preceding charge. Thus, if 8 skip-loads make a complete charge, the first ore of one charge will be dumped at a point 32 deg. from that of the first ore of the preceding charge, and so on. It is this feature of the distribution which makes the apparatus valuable.

FIG. 10.



A DISTRIBUTOR AND A DOUBLE SKIP.

The breakage of coke or ore is relatively small; for the hopper-chute is kept as close to the main hopper as possible, and this distance may be cut down to less than 10 ft.

This charger was patented before the mechanical filling of blast-furnaces became the general practice; and the furnace managers of that day, while recognizing almost universally the theoretical value of Brown's invention, thought it involved too much mechanism for a blast-furnace; and the double bell, by virtue of its simplicity, became the most common type. Now, however, the serious defects of the double-bell charger have been demonstrated, while on the other hand Brown's device has been simplified. Fig. 10 shows its present form for the single skip, with the double bell-rod. The whole of the gas-seal and distributor is on wheels, and can be jacked up and run to one side when repairs must be made to the main bell-and-hopper.

The gas-seal door is open, except when the bell opens. Then the first three inches' travel of the main bell allows the door to drop shut, and the top is sealed. During charging, the door, being open, allows the stock to flow freely out of the chute and gives a more uniform distribution in the main hopper.

Fig. 11 shows the distributor supplied with the double skip. No superstructure is required over the top of the furnace except what is necessary to handle the bell for repairs.

Fig. 12 shows the revolving hopper or chute removed from the gas-seal cage and support.

Fig. 13 shows the gas-seal cage, which is made of reinforced steel plate in this case, but is sometimes made of cast-iron when preferred.

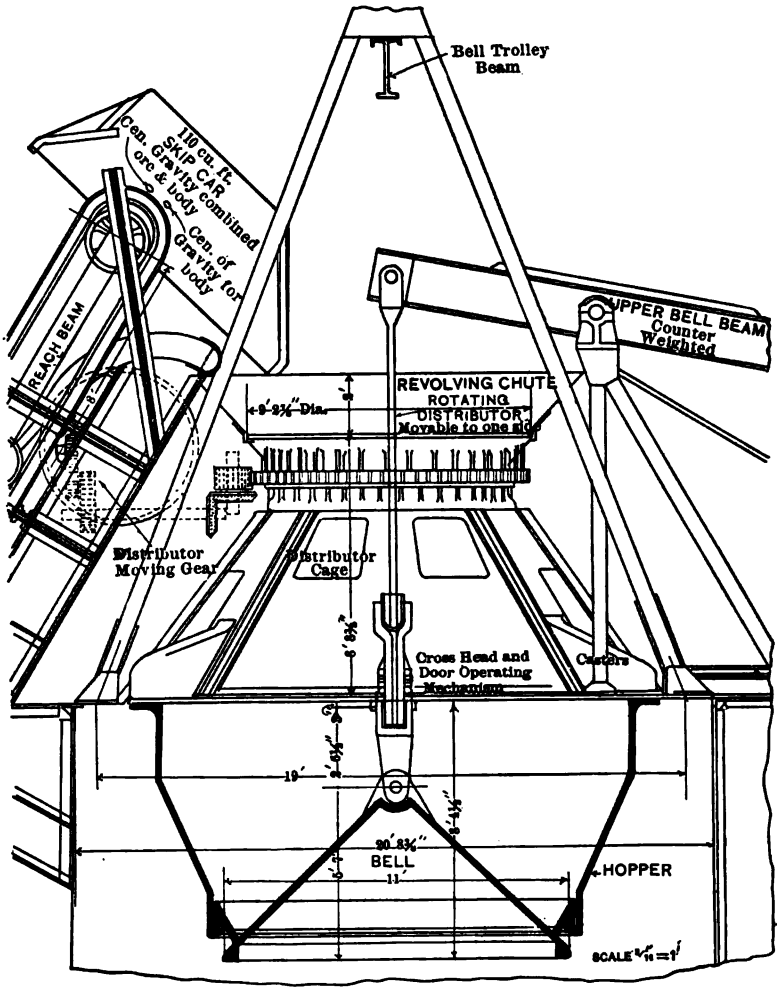
In the earlier forms the revolving hopper was supported on wheels, and later, on steel balls. In the present form all these are dispensed with, and the flange of the hopper simply slides at the top of the gas seal, on a surface of cast-iron against cast-iron, about 8 in. wide, and lubricated only with graphite. This arrangement, contrary to prediction, has been found to show practically no wear after a year's use. If they will last through a blast, the arrangement will have served its purpose well; and it seems, under actual trial, to be entirely capable of that degree of durability.

The flue-dirt does not seem to enter the joint to any extent.

I believe the reason lies in the fact, already noted, that the gas-seal door is only closed when the main bell is open.

The pressure on the sliding surfaces is only about 10 lb. per sq. in., and the friction under that load cannot be great.

FIG. 11.



SIDE ELEVATION

A DISTRIBUTOR AND SINGLE SKIP WITH A DOUBLE BELL-ROD.

This distributing-top has passed beyond the experimental stage and has shown itself in practice to possess the advantages, without the disadvantages, of hand-filling. In all the

FIG. 12.

TWO SIDE VIEWS OF DISTRIBUTOR, AT RIGHT ANGLES TO EACH OTHER.

FIG. 13.

SIDE VIEW OF GAS-SEAL CAGE.

furnaces where it has been installed, it has reduced the slips and prevented the irregular wear of the in-wall. There are indications that it will show a marked saving in fuel; but the data for comparison in this respect are not yet sufficiently complete for publication.

If the present prospect of technical efficiency should be confirmed by longer use, only one further improvement will be required, to secure results superior in economy to those obtained by hand-charging—namely, an arrangement of bins and apparatus for handling coke which will do away with the excessive production of dust involved in present methods of storage and rough handling and re-handling of this fuel. This subject has been treated in my paper on "Stock-Distribution," etc., already cited.

Special Forms of Blast-Furnace Charging-Apparatus.

BY T. F. WITHERBEE, DURANGO, MEXICO.

(Lake Superior Meeting, September, 1904.)

THAT the single charging-bell, properly proportioned, is a good all-around device for distributing material in a blast-furnace can scarcely be questioned; yet it is equally true that, in some cases, it has not given perfect satisfaction; as has been demonstrated by the substitution of double bells, at furnaces under the best management—generally in cases where anthracite, either alone, or with a portion of coke, was used as fuel. The defect of the single bell seems to be that it does not permit, beyond very narrow limits, such changes in charging as may be required by the temporary condition of the furnace.

Modern furnaces using Connellsville or other good coke and Lake Superior ores seem to be well served with a single bell. At least, up to a certain size, somewhere about 18 by 85 ft. to 19 by 90 feet, these furnaces make low-silicon pig-iron with about 1,800 lb. of fuel, or in some cases very much less.

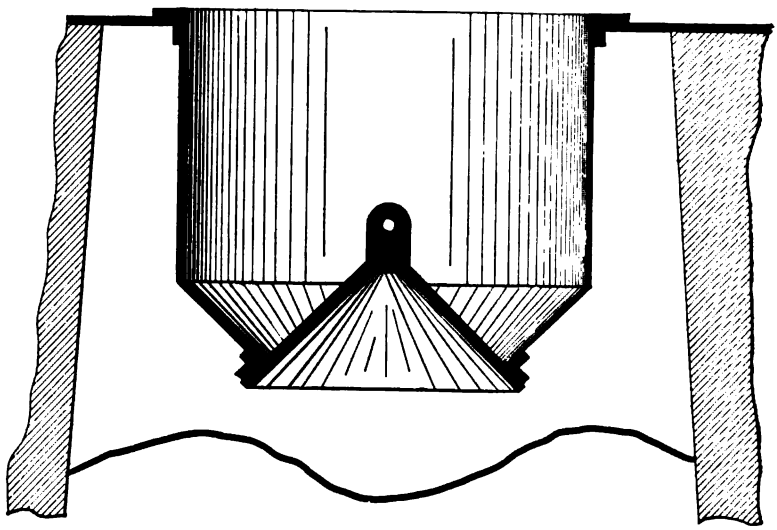
The following instances occur to me, in which other forms of charging-apparatus have been employed with advantage:

1. A charcoal-furnace, 61 ft. high, 11 ft. in diameter at bosh, and 8 ft. 2 in. at stock-line, with a rectangular crucible, 42 by

36 in. in size, and bell-and-hopper as shown in Fig. 1, yielded, at the start, black scouring cinder and white iron, and so continued for 30 days, when it was shoveled out. The gas-circulation had been wholly through the center; and for 30 ft. above the tuyeres no fire or heat had reached the walls, the red color of the fire-clay still showing in the fire-brick joints. It is probable that the small crucible, only 36 in. wide, had something to do with the center circulation. This feature was unavoidable, since the old stone stack would not permit a larger hearth.

To remedy this trouble, the charging-apparatus shown in Fig. 2 was designed.

FIG. 1.



Distributes the charge in a ring next to the lining.

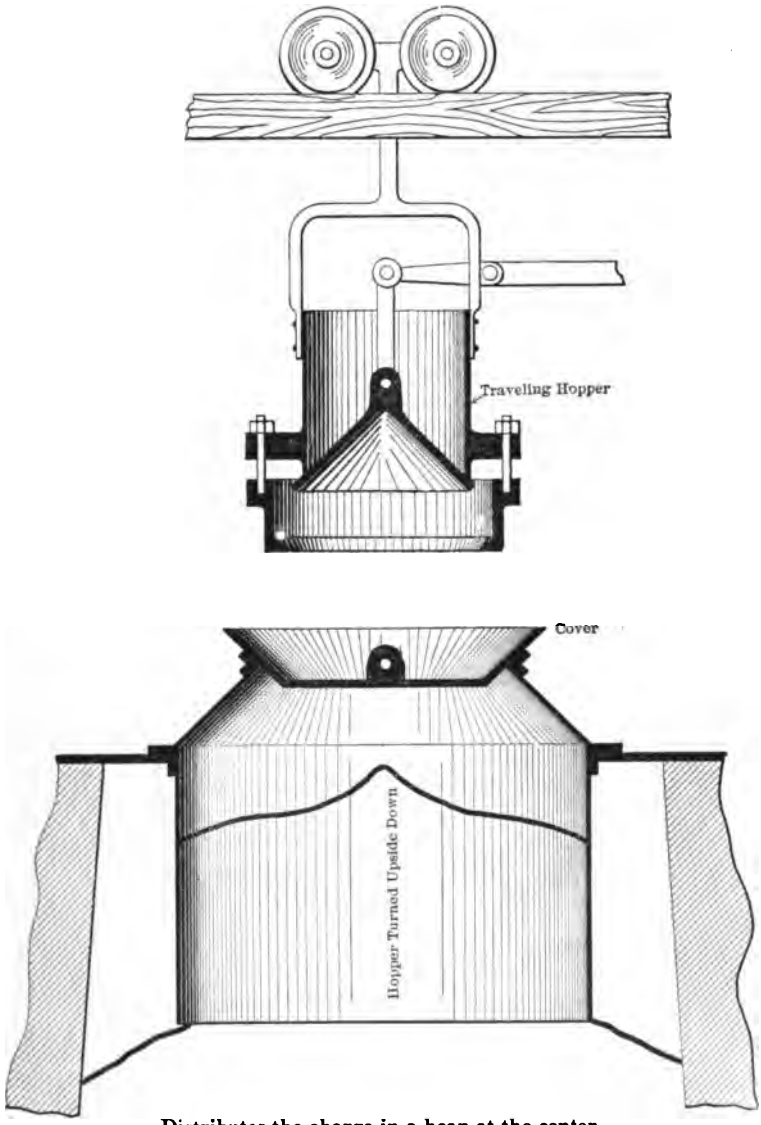
SINGLE BELL AND HOPPER AT A CHARCOAL IRON FURNACE.

This deposited all the ore and stone in a heap about 3 ft. in diameter in the middle of the furnace. The result was favorable, except that at intervals of about eight or ten days, scouring cinder and no iron would be made for two or three (in one instance for six) hours, after which regular work would begin again. That trouble was cured by increasing each fuel-charge from 20 bushels of charcoal to 100 of charcoal and 25 of wood.

The fuel-consumption averaged for considerable periods less than 1,600 lb., and even for long campaigns 2,000 lb. of fuel

per ton of 2,300 lb. of iron. Considering that such work was done with Champlain magnetites and iron-pipe stoves more

FIG. 2.



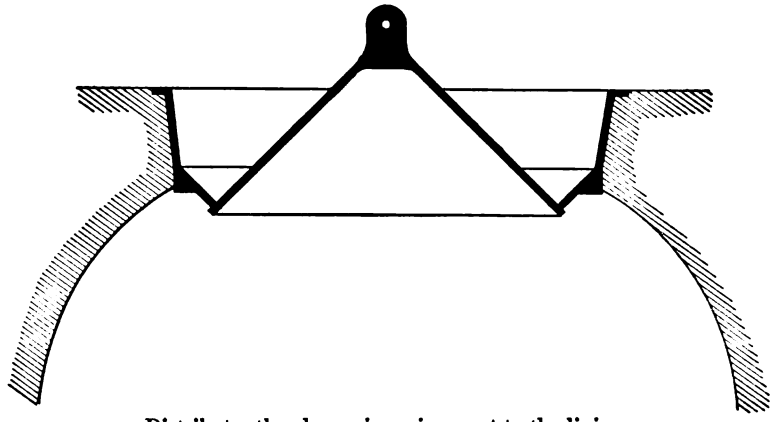
Distributes the charge in a heap at the center.

LATER FORM OF SINGLE BELL AND HOPPER AT A CHARCOAL IRON FURNACE.

than 30 years ago, according to the practice and knowledge of that period, we must admit that the results were at least fairly

good, and proved such a system of charging to be not incompatible with fuel-economy.

FIG. 3.

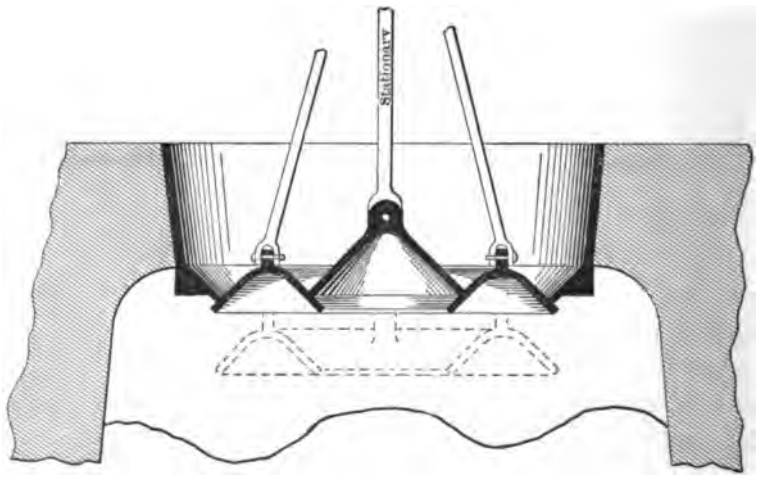


Distributes the charge in a ring next to the lining.

SINGLE BELL AND HOPPER AT AN ANTHRACITE IRON FURNACE.

This apparatus was only a make-shift; but so satisfactory was its working that it was used for about 5 years (until the

FIG. 4.



Distributes the charge both in a ring next to the lining and in a heap at the center.

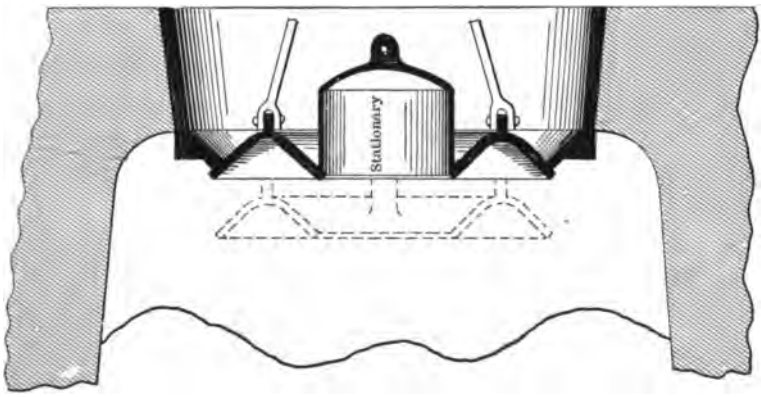
THE BAUMAN BELL AND HOPPER.

charcoal-supply became exhausted), during which time the furnace worked smoothly without a single slip.

As was usually the case in those days, the blowing-power was inadequate, and experience soon demonstrated that when the diameter of the tuyeres was increased beyond 1.75 in. for each of the six tuyeres, penetration was lost, and white iron was produced. In this case the resistance was in the tuyeres and not in the stock.

2. A second instance of unsatisfactory working with a single bell was at an anthracite-furnace 71 ft. high, 16 ft. in diameter at bosh, 14.5 ft. at stock-line, and 8 ft. at the crucible, with six 4-in. tuyeres, and a 7.5 ft. bell as shown in Fig. 3. Notwithstanding the relatively large stock-line diameter, the bell banked the ore snugly against the furnace-lining.

FIG. 5.



Distributes the charge both in a ring next to the lining and in a heap at the center.

FIRMSTONE'S MODIFICATION OF THE BAUMAN BELL AND HOPPER.

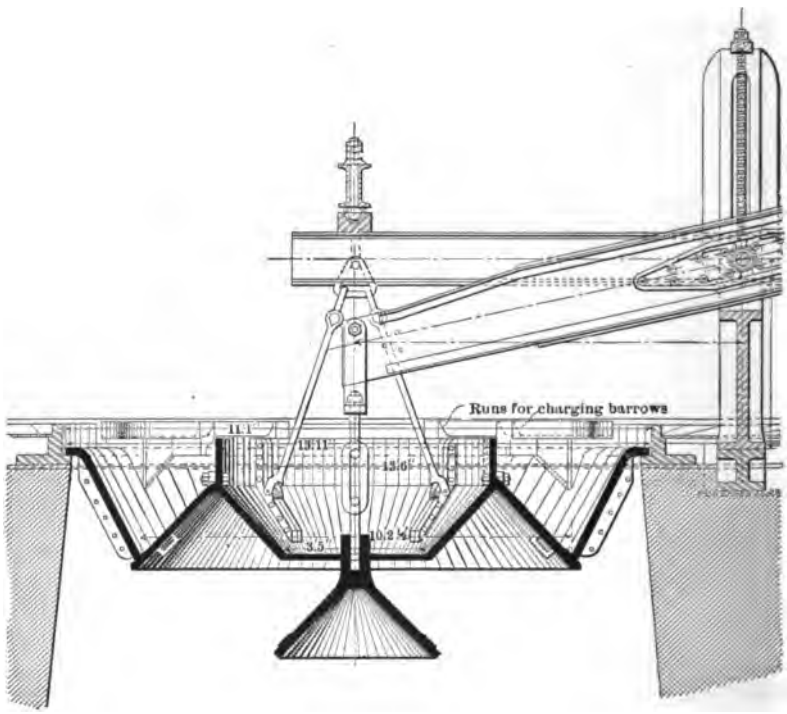
A good blow-in was effected, but at the end of three months the furnace got into a scrape and the campaign was finally (though, as subsequent experience demonstrated, unnecessarily) abandoned. Examination showed a hole 55 ft. deep, burned through the center, leaving about 4 ft. of loose stock on the walls. The furnace could have been saved by filling the hole with anthracite, and applying an extra volume of blast.

Subsequent campaigns, so long as anthracite only was used, showed a tendency to center-circulation, although the stock-line diameter was reduced to 12 ft. Indeed, why should this not be the case, especially when dense, heavy, fine magnetite is

charged? The natural effect of a single bell, at least under the conditions above stated, is to render the outside ring of materials more impervious to the ascending gases than the center, because only the large pieces of fuel and stone roll to the center, while the ore stays about where it falls.

This tendency to burn up through the middle was, however, very much reduced by using coke as a part of the fuel. The

FIG. 6.



Distributes the charge in three ways:—1. In a large ring next to the furnace lining. 2. In a small ring. 3. In both a large and a small ring.

THE DURHAM BELL AND HOPPER.

more rapid working thus effected may have been the secret of this improvement in results.

Later, the single bell was replaced with the Bauman bell shown in Fig. 4, and afterwards Firmstone's modification of the Bauman (see Fig. 5) was added. This improved design was used at Glendon, Pa., a long time, and therefore, presumably, with satisfactory results.

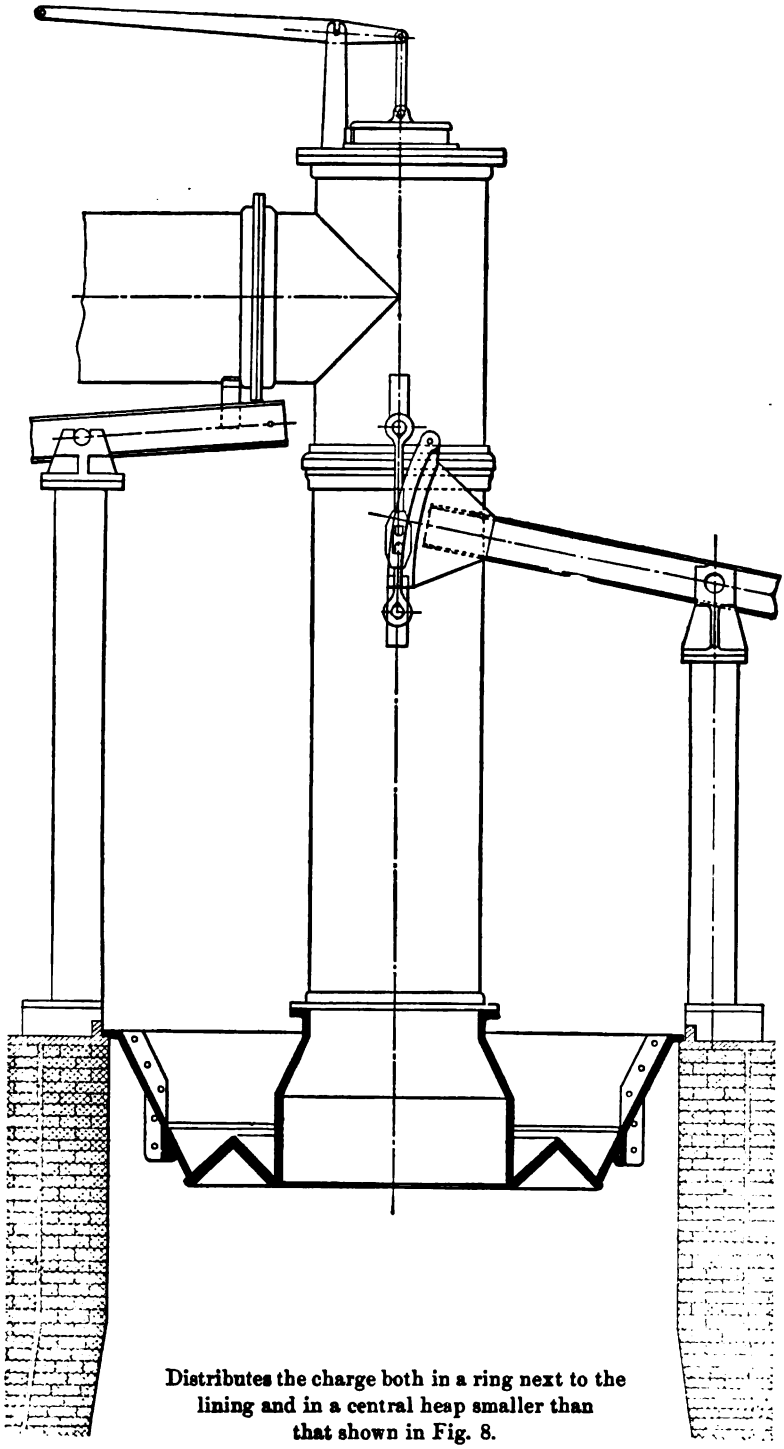
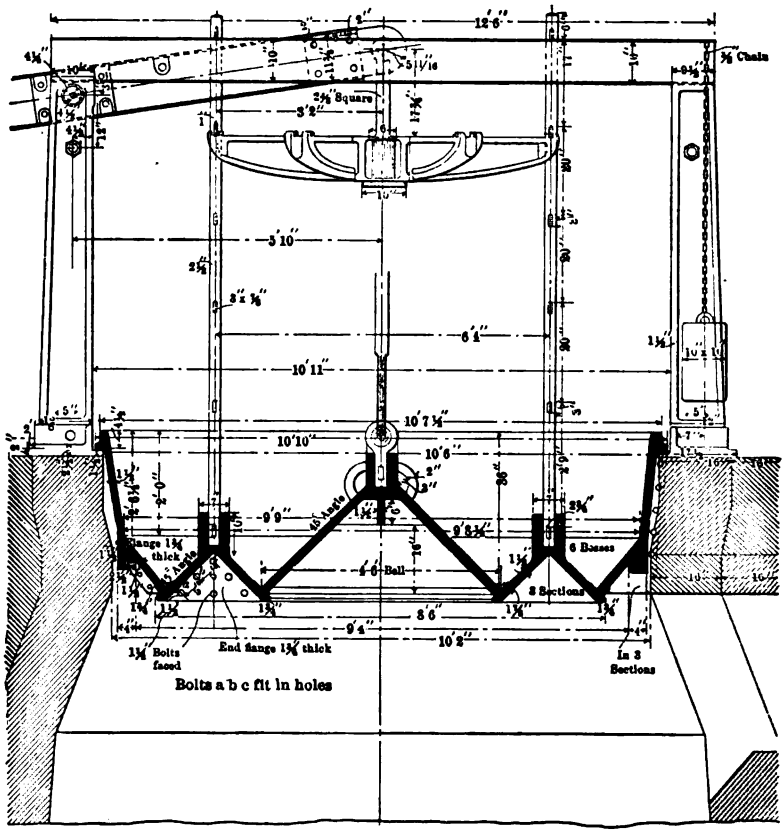


FIG. 7.—THE BAUMAN-FIRMSTONE BELL AND HOPPER AT THE LONGDALE FURNACE, VA.

3. Fig. 6 represents the Durham bell,[1] the invention of Mr. Edward Cooper, who made many interesting experiments with it. In this design the hopper on the annular bell gives control of the distribution. This was, I believe, the first device which embodied that important feature; and for this and other valu-

FIG. 8.



Distributes the charge both in a ring next to the lining and in a heap at the center.

BAUMAN DOUBLE BELL AND HOPPER AT THE SAUCON FURNACE, HELLERTOWN, PA.

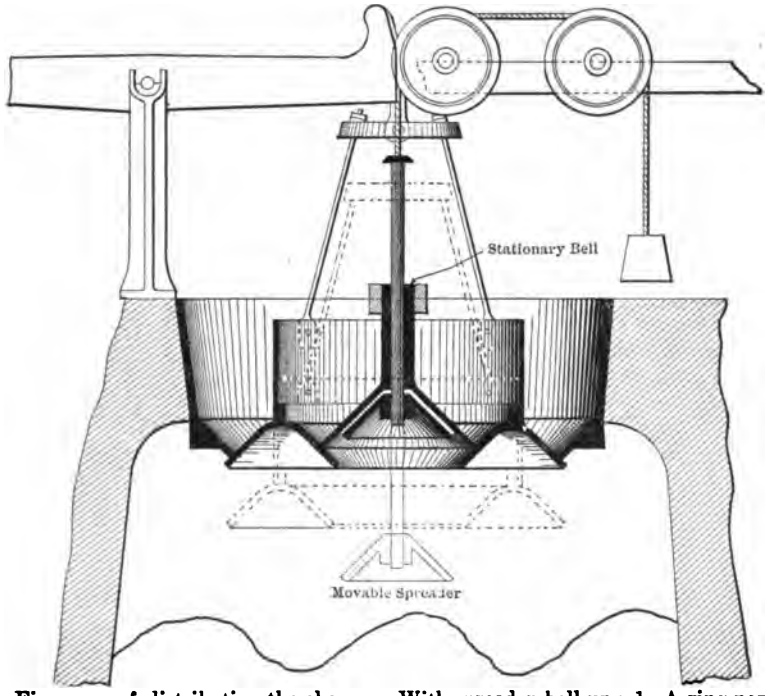
able contributions to technical progress (notably the Durham iron stove) Edward Cooper deserves the hearty thanks of American ironmasters, especially because he has always generously placed the results of his study and ingenuity at the disposal of his professional colleagues, never seeking to hinder the free use of them by protecting them with patents.

[1] *Trans.*, xiv., 136.

4. A charging-apparatus used at Longdale furnace, Va., is shown in Fig. 7. It is practically a Bauman-Firmstone, with provision for taking the gas off in the center—a feature which has been highly commended.

5. The double bell of the Bauman type, shown in Fig. 8, has been used by the Saucon Iron Co., Hellertown, Pa. It will be noticed that the Bauman bell deposits the ore in an annular

FIG. 9.



Five ways of distributing the charge.—With spreader bell up: 1. A ring next to lining. 2. A center heap. 3. A ring next to lining and a center heap. With spreader bell down: 4. A large and a small ring. 5. A small ring.

THE WITHERBEE DOUBLE BELL AND HOPPER.

ring and also in a heap at the center, while the Durham bell deposits it in a large or small annular ring, or both, according as the inner or outer hopper is used.

6. In 1887 I designed the bell shown in Fig. 9, which, by means of the suspended spreader-bell, can be used to effect the same distribution as the Bauman or Durham, according as the "spreader" may be turned up or down.

This apparatus was intended to meet special conditions, particularly the coming use of Champlain magnetic concentrates, which were then as fine as Mesabi ores, besides having a higher specific gravity, and being, therefore, more inclined to run. It was under my charge for only five months, during which the furnace worked normally, and the special efficiency of the apparatus was only negatively demonstrated. As an experiment, however, the furnace was charged for about a week wholly through the inner hopper, with spreader raised, all the stock being thus put in a heap in the center. Normal work continued; but the lining came up 3 in., thus demonstrating, what I had expected, that by means of this apparatus the ascending gases could be diverted and controlled at will.

Mr. N. M. Langdon, a member of the Institute, completed the blast with this charger; and when I saw it next, after my return from Mexico in 1888, it was lying on the stock-house floor, badly burned and warped (probably in blowing-out), and it was subsequently replaced with a single bell. Mr. Langdon told me that he had discovered no particular difference between it and a single bell (which, perhaps, was commendation enough); but I am still of the opinion that the Bauman, the Durham, and the combination of the two as shown in Fig. 9, have some merit, at least for special cases. Some may consider them complicated as compared with a single bell; but in practice they are not so, since one valve controls their manipulation.

It is currently reported that the modern large furnaces (from 90 ft. up) have not proved very satisfactory, so far as regularity of working and fuel-economy is concerned. In the West, several have been blown out, after a few weeks' run, for decapitation and other changes; and the manager of one of the largest groups of furnaces in the United States predicted that the two which his company had under construction would not surpass the stacks 85 by 18 ft. in size, except in quantity of product. The performance of these two new furnaces has confirmed his judgment, since they have never consumed less than 2,000 lb. of fuel per ton of iron. It seems incredible that an addition of a foot or two to bosh-diameter, and 15 or 20 ft. to height, should necessarily be attended with such unsatisfactory results as are

reported. One of my informants attributes this experience to a "wandering" of the blast—which is very possible, since the conditions for such a wandering are provided by the (from one point of view) excessive tuyere-area.

At an anthracite-furnace, with blast heated to about 1,400° F., the tuyere-pipes were, of course, generally bright red; but it was noted occasionally that only two or three of the six tuyeres were so, the others being black, showing that nearly all the blast was passing through the red-hot ones. By partly closing the valves in the tuyere-stocks of the hotter ones, the others could be reddened, and the blast, thereby, more evenly distributed.

Subsequently, a direct experiment showed that the volume of blast which was entering the furnace through 75 sq. in. of total tuyere-area, would pass out into the open air through 30 sq. in. at the same pressure, proving that the resistance was in the stock, and not in the tuyere. Probably this condition exists approximately at all blast-furnaces. Many years ago, Mr. John M. Hartman invented a device to show at a glance the distribution of the blast. As I remember it, it throttled the blast at each tuyere-stock, thus maintaining a slightly higher pressure in the bustle-windpipe than in the furnace, so that a blast-pressure gauge, applied to each tuyere below the throttling-diaphragm, would show the pressure in each tuyere. If this was equal to the bustle-pipe pressure, it indicated an obstruction in the zone of that particular tuyere.

While it is easy to demonstrate blast-wandering, the remedy is not so apparent. Maintaining a considerably higher pressure in the bustle-pipe may tend to remedy the evil; but so long as the method of charging keeps the center relatively more open, it will not be practicable to reduce the tuyere-area; for, in such a case, the open center would be burned out, and the most serious of all blast-furnace derangements, a ring-scaffold, would be set up, with its accompanying dust-throwing, slips and so-called explosions.

Perhaps any defects in working, which may have been shown by the "Jumbo" furnaces, are due, not to the size of the stacks, but to the way in which the materials, including the blast, have been distributed. If their reported fuel-consumption could be brought down to the best records of the smaller furnaces, it

would represent a saving of more than $100,000 per annum for each furnace—a sum of sufficient importance to justify the slightly-increased complication of the charging-apparatus.

It will be a surprise if the monster Buffalo furnaces do not even surpass this fuel-economy—not on account of any extra super-heated blast that may be at their command, but by reason of their improved charging-apparatus, which, to my mind, shows a step in the right direction.

It is to be understood that whatever reference has been made above to large furnaces applies to a few specific cases, reported to me, which may not be typical ones. Moreover, in these cases, fuel-economy may have been sacrificed for quantity of product, as is sometimes intentionally done for good commercial reasons.

The main purpose of this paper is to present the suggestion, that better results in running might be had by charging nearer to the center, so as to have a more resistant column of materials there to blow against, thereby enabling the reduction of the tuyere-area enough to minimize blast-wandering and its train of evils.

As the matter now stands, we are obliged to use, with large bells, tuyeres from 50 to 60 per cent. too large, and thus practically to surrender all control of the blast at the bustle-pipe, from which it can enter the furnace through any tuyeres offering a relatively smaller resistance.

I am under obligations to Messrs. Frank Firmstone, B. F. Fackenthal, Jr., and David H. Thomas, for drawings and valuable data embodied in this paper.

Crushing in Cyanide Solution, as Practiced in the Black Hills, South Dakota.

BY CHARLES H. FULTON, SOUTH DAKOTA SCHOOL OF MINES, RAPID CITY, SO. DAK.

(Lake Superior Meeting, September, 1904.)

INTRODUCTION.

THE process of crushing ore in cyanide solution was first used at the Crowns mine, New Zealand, in 1897, by Mr. F. R. W. Daw; and, two years later, Mr. John Hinton tried it experimentally at the old Dakota plant, Central City, South Dakota. To Mr. Hinton belongs the credit of introducing this important modification in the Black Hills, this being its first successful application in the U. S., and its results have been so satisfactory that it has become established as the proper manner in which to treat the dense oxidized siliceous ores of this district. Five mills are now crushing ore in cyanide solution, namely, the Horseshoe, 120 stamps (60 in operation); the Dakota, 30 stamps; the Maitland, 40 stamps; the Hidden Fortune, 60 stamps; the Lundberg, Dorr & Wilson, one 6-ft. Monadnock roller-mill; and several others intend to adopt this practice. The dry-crushing process still holds its place for the treatment of the more porous and open siliceous ores; and two mills continue to dry-crush dense siliceous ores.

CHARACTER OF THE ORES.

The occurrence of the ores of the Cambrian strata in the northern Black Hills has been frequently described.[1]

In practice they are classified, according to their appearance, into "red" and "blue" ores, the former being oxidized and the latter unoxidized. At the present time, usually the red ores only are treated at the mills, although occasionally small quantities of blue ore, which have become intimately mixed with the red in mining, are included. Generally the extraction of values by the direct treatment of raw ore is from 70 to 75 per cent. for red ores, and from 25 to 50 per cent. for blue ores. The ores consist mainly of a siliceous matrix containing, as a

[1] A Contribution to the Geology of the Northern Black Hills, J. D. Irving, *Annals of the New York Academy of Sciences*, vol. xii., p. 187–340 (1899).

rule, from 6 to 8 per cent. of pyrite and, sometimes, a little arsenopyrite. The oxidized ores, from which the sulphur has been eliminated, contain the iron mostly in the form of limonite. Very little is known concerning the manner in which the gold occurs. Practically only 1 or 2 per cent. of the gold can be recovered by amalgamation, even in the oxidized ores.

In the analysis given in Table I. particular attention has been paid to tellurium, antimony, arsenic, copper, and such other elements as are present in minute quantities only, or might be present in a complex mineral form carrying the gold- and silver-values. It was thought that gold might occur in most of the ores as a telluride; but experimental investigation giving negative results, renders this supposition extremely doubtful. Extensive experiments in the treatment of these ores by cyanide solutions have been made at the South Dakota School of Mines; and the results show that a portion of the values are practically insoluble in cyanide solution, irrespective of the fineness to which the ore has been reduced, the extent of the roasting, or the time of the treatment, aside from the addition of bromocyanogen and similar reagents. The conclusions deduced were, that the values were probably in combination with a complex mineral containing arsenic, antimony and, perhaps, bismuth.[2]

TABLE I.—*Analyses of Typical Red and Blue Ores from the Black Hills, So. Dak.*

	Blue Ore, Maitland Mine.	Red Ore, Maitland Mine.	Dense Siliceous "Blue" Ore, Yellow Creek District.
	Per Cent.	Per Cent.	Per Cent.
Silica	77.38	70.95	93.72
Iron	3.54	10.30	2.67
Sulphur	4.42	1.66	0.69
Arsenic	0.55	0.30	0.02
Antimony	trace	trace	0.0393
Tellurium	none	0.002	none
Copper	trace	0.02	none
Manganese	none	trace	0.082
Alumina	2.80	4.30	3.53
Lime	0.56	3.40	trace
Magnesia	trace	1.02	none
Phosphoric acid	0.32	none	0.0059
Soda	1.32	.....	none
Lead	trace	.....	trace
Thallium	?	.....	?
Tungsten	none	.....	none
Gold, oz. per ton	0.78	0.56	0.90
Silver, oz. per ton	1.00	1.03	.....

[2] Laboratory Experiments on the Unoxidized Siliceous Ores of the Black Hills, *Bulletin No. 7, South Dakota School of Mines* (June, 1904).

The analyses in Table I. represent very closely the average composition of the siliceous ores, although in some of the more shaly ores the percentage of alumina is considerably greater and that of silica correspondingly lower; also the percentages of sulphur and iron are sometimes greater. Physically considered, the ores may be divided into "compact quartzite" ores and "shale" ores. The latter class yields a slime much more troublesome to treat. Both classes of ore are treated at some of the mills in the district.

General Features of the Process.

The process comprises the following operations:

1. Crushing of the ores, generally by stamps, in a weak cyanide solution ranging from 1.3 to 2.2 lb. of KCN per ton, and having a protective alkalinity, equivalent to from 1 to 1.5 lb. of sodium hydrate per ton.
2. Separation of the sands from the slimes by means of cone-classifiers.
3. Treatment of the sands by percolation.
4. Treatment of the slimes by agitation and decantation.
5. Precipitation of the gold- and silver-values by means of zinc thread.

This process is applicable to the dense siliceous ores which require comparatively fine crushing and which contain but a small quantity of cyanicides. It is not applicable to ores which destroy much cyanide, unless they are previously treated by an alkaline wash; and the consumption of cyanide under all cases is greater than in the dry-crushing process. In the wet-crushing mills of the district, the quantity of cyanide consumed per ton of ore treated varies from 0.75 to 1.50 lb., while at the Imperial mill, at Deadwood (a typical dry-crushing plant), treating the same class of ore, the consumption is 0.5 lb. per ton; and at the other dry-crushing plants of the district, it ranges from 0.4 to 0.75 lb. per ton. The increased consumption of cyanide is a defect which is due to the following inherent causes. (1) The agitation of the ore with cyanide solution in the battery causes an extra consumption of cyanide. (2) Although the battery cyanide solution has a protective alkalinity (above that due to cyanide and cyanogen compounds) of from 1 to 1.5 lb. NaOH per ton, this does not, by any means,

completely protect the cyanide from decomposition by cyanicides; the reaction between cyanicides and cyanide, and alkaline-earth hydrates and caustic alkalies, probably taking place, to some extent, simultaneously. Metallurgists recognise that it is frequently essential to use a comparatively highly alkaline solution containing but a small quantity of cyanide on the ores, before the stronger cyanide solutions are applied, for the reason that the protective alkalinity of the strong cyanide solution would not be effective in preventing a considerable decomposition of cyanide. (3) There is also an increase in the consumption due to the discharge of much cyanide in the moisture passing out with the slimes-tailings, which might be called a mechanical consumption. The quantity of cyanide consumed in this manner alone, amounts to from 0.3 to 0.6 lb. per ton of ore treated. The mechanical consumption of cyanide in a dry-crushing plant is so small as to be negligible.

At the present time, it is difficult to make a comparison of the relative merits of crushing ore in cyanide solution and dry-crushing. On the whole, there is probably little difference in the respective costs, although wet-crushing is slightly cheaper, in spite of the slimes-treatment and the higher consumption of chemicals involved; and of course is free from the dust-nuisance. The hope that, on account of the greater fineness of crushing permissible, the wet-crushing plants would be able to treat successfully the blue ores in the raw state, has not been realized; and it is probable that these ores will finally have to be roasted, which is advantageous for dry-crushing. Some of the mines of the district furnish but little blue ore, while others have a great deal in their ore-reserves. For the treatment of red ores, crushing in cyanide solution is, without doubt, firmly installed.

Crushing the Ores.

The ores are, as a rule, roughly broken in a Gates breaker, although, at the Maitland mill, a Blake breaker is used for this purpose. The broken ore, fine enough to pass through a 1.5- or 2-in. ring, is fed by Challenge ore-feeders to the stamps, which, with one exception, are used for the fine-crushing of the ore. The Lundberg, Dorr & Wilson mill uses a 6-ft. Monadnock roller-mill for the fine-crushing, a set of rolls being placed

between the Gates breaker and the rolls, in order to get the proper-sized feed of ore.

Details of the Stamp-Mills.—Table II. gives the details of the stamps at four of the mills in the Black Hills district.

TABLE II.—*Details of Stamp-Mills in the Black Hills.*

Name of Mill.	Weight of Stamp.	Number of Drops per Minute.	Height of Drop.	Depth of Discharge.	Width of Mortar Box at Discharge-Level.	Screen Used.	Amount of Solution Used per Ton of Ore Crushed.	Capacity per Stamp per 24 Hours.	Type of Mortar.
	Tons.		In.	In.	In.		Tons.	Tons.	
Hidden Fortune.*	1,020	98	7	8	16	24-mesh, No. 26 wire.	5	4	Double discharge, rear blocked up.
Maitland...	910	97	7 to 8	6	13	26- by 13-mesh, No. 26 wire.	4 to 5	2.5 to 3	Single discharge.
Dakota.....	950	88	8	9	15	10- by 4-mesh, No. 20 wire.	5	4.4	Single discharge.
Dakota.....	950	88	8	7	22	10- by 4-mesh, No. 20 wire.	5	4.4	Double discharge, rear blocked up.
Horseshoe..	1,000	90	8	5 to 6	18	14- by 7-mesh, No. 21 wire.	6	4.5 to 5	Double discharge, rear blocked up.

* Crushing in cyanide solution containing 1.3 lb. KCN per ton; and amalgamating both inside and outside of the mortar.

Some of the earlier mills installed double-discharge mortars, hoping to get an increased stamp-duty, but it was soon found that the quantity of solution required in the crushing was too great for economical handling, and the rear discharges were then closed by wooden frames. It will be seen from the data in Table II. that the depth of discharge and the width of the mortar at the discharge-level varies considerably at the different mills, while the variation in the weight of the stamp is not so great.

At all of the mills, except the Hidden Fortune, the stamps used are for crushing-purposes only; and generally a narrow box, a shallow discharge and a heavy stamp (weighing from 1,100 to 1,200 lb.) seems to give the greatest capacity and the most economy. In this connection, however, it must be borne in mind that it is desirable to retain the ore in the mortar for a certain length of time, in order to agitate it thoroughly with cyanide solution. In most cases, from 50 to 53 per cent. of the precious-metal values of the ore are extracted in the batteries and classifying-cones. Stamps for this kind of crushing

should have a daily capacity of at least 5 tons per head; and they have been designed accordingly in the later plans of prospective mills. Some of the mills treat very hard ore, which accounts, in part, for their relatively low capacities.

The size of the screens used at the different mills ranges from 26-mesh, No. 26 wire, to 10-mesh, No. 20 wire,—a variation called for by the requirements of the individual ores, which, although of the same general characteristics, differ somewhat in the fineness of crushing needed for a proper extraction of the gold and silver contained in them.

Several of the mills have recently installed wire-cloth screens in which the opening is rectangular, instead of square, the longer dimension being from 2 to 2.5 times greater than the shorter. Screens of this type give a slightly greater capacity and do not choke as readily as screens having a square opening.

The denser siliceous ores require to be crushed comparatively fine, but, if crushed too fine, nothing is gained in extraction, and trouble is caused by the production of an excessive quantity of slimes which is difficult to handle in the mill. The most economical extraction is obtained if the ore is so crushed that the size of the greater bulk of it is between 30-mesh (0.0195 in.) and 60-mesh (0.0075 in.). Material of a size finer than 60-mesh yields but a slightly higher extraction than that between 30- and 60-mesh. Although the recovery of values from sands and slimes is the same, the 5-per cent. greater extraction obtained from the slimes is due mainly to the agitation. However, if the size of the ore-particles is coarser than 30-mesh, the extraction from most of the ores is materially decreased.

Table III. shows the nature of the mill-products at some of the mills.

The Dakota mill uses the coarsest screen of all the mills and gets a product of which about 20 per cent. is coarser than 30-mesh size. This proportion is rather large, but in view of the very low gold-tenor of the ores treated, and their shaly nature, crushing in this manner is the most economical. The other mills use finer screens, and the sizes of their mill-products approximate that of the Monadnock mill quoted in Table III. The Hidden Fortune mill crushes some "cement" ore, which

TABLE III.—*Mechanical Analyses of Mill-Products.*

Mesh of Screen. Ordinary Brass-Wire Assayers' Screen.	Name of Mill.			
		Dakota. Stamps.Screen 10- by 4-Mesh, No. 20 Wire.	Dorr & Wilson. Monadnock Roller-Mill. Screen, 18-Mesh, 0.046 in. Space	Imperial. Dry-Crushing Rolls. Screen, 16-Mesh, No. 21 Wire.
		Per Cent.	Per Cent.	Per Cent.
Plus 20-mesh.	Percentage remaining on screen.	12.7	1.0	3.0
Plus 30-mesh.		.....	.....	18.0
Plus 40-mesh.		22.0	10.0	17.0
Plus 60-mesh.		.....	10.0	16.0
Plus 80-mesh.		14.8	.....	5.0
Plus 100 mesh.		6.0	10.0	5.0
Minus 100-mesh.		.....	.....	36.0
Plus 150-mesh.		14.8	.....	.....
Minus 150-mesh.		29.8	.....	.....
Plus 200-mesh.		....	19.0	.....
Minus 200-mesh.		.....	50.0	.....

contains a considerable percentage of coarse free gold. This mill now crushes the ore in cyanide solution and amalgamates both inside and outside of the mortar. A very weak cyanide solution (1.3 lb. of KCN per ton) is used successfully in the battery, and no difficulty is experienced in getting a good amalgamation. This method is used in part at one or two mills using the Diehl process at Kalgoorlie, Australia.[3] The plates require to be dressed more frequently, owing to the hardening action of the cyanide on the amalgam; and it is very probable that they will have to be renewed more frequently on account of the solvent action of the cyanide.

Most of the mills now use chrome-steel for the shoes and, in part, for dies, for the reason that it gives the most satisfaction and is the most economical. Table IV. gives a comparison of the cost and the working-results of shoes and dies made of different materials.

The Dakota mill, also, uses chrome-steel shoes and is now experimenting with a cast-iron die made at a local foundry from a mixture containing 20 per cent. of chrome-steel scrap. The product costs 3.5c. per lb. laid down at the mill. Dies of this material weighing 120 lb. have lasted 46 days, during which time 175 tons of ore have been crushed on them, leaving 24 lb. of scrap metal, which was sold at 0.5c. per lb.

[3] The Diehl Process, H. Knutzen, *Transactions of the Institution of Mining and Metallurgy* (London), vol. xii., pp. 1 to 23 (1902).

TABLE IV.—*The Cost and the Working-Results of Shoes and Dies of Different Materials.*

Name of Mill.	Name of Part.	Material.	Weight of Ore Crushed by One Shoe or Die.	Number of Days Used.	Cost, per Ton of Ore Crushed.
			Tons.		Cents.
Maitland.....	Shoe.	Chrome-steel.	250	95	4.46
Maitland.....	Shoe.	Cast-iron.	105	40	4.95
Maitland.....	Die.	Cast-iron.	105	40	3.28
Maitland.....	Die.	Wilson steel.	280	105	3.06
Horseshoe...	Shoe.	Chrome-steel.	336	84	3.12
Horseshoe...	Shoe.	Cast-iron.	104	26	6.15
Horseshoe...	Shoe.	Wilson steel.	280	70	3.67
Horseshoe...	Die.	Chrome-steel.	400	100	2.29
Horseshoe...	Die.	Cast-iron.	120	30	4.08
Horseshoe...	Die.	Wilson steel.	340	85	2.35

NOTE.—Weight of shoes, 180 lb.; and dies from 120 to 140 lb. Chrome-steel, laid down at Terry, So. Dak., costs 5.83c. per lb., Wilson forged steel, 5.72c. per lb., and cast-iron 3.5c. per lb.

At the Lundberg, Dorr & Wilson mill, a 6-ft. Monadnock roller-mill is used in place of stamps to crush the ores in the cyanide solution. This mill crushes about 90 tons of ore per day, from 0.75-in. size through an 18-mesh screen, having a space of 0.046 in. square. The mill makes 32 rev. per min. and has 19.5 sq. ft. of screen-area. A peculiar feature of this mill is that both the die-ring and the roller-tire cup, instead of the die cupping and the tire crowning. This result, however, does not seem to affect the efficiency of the crushing. The mill is giving satisfaction, but, as yet, no data of results in the crushing of siliceous ore are available, with which to make a comparison between its work and that of stamps. The ores crushed at the Lundberg, Dorr & Wilson mill are in part comparatively soft, although some hard blue quartzite-ores also are crushed.

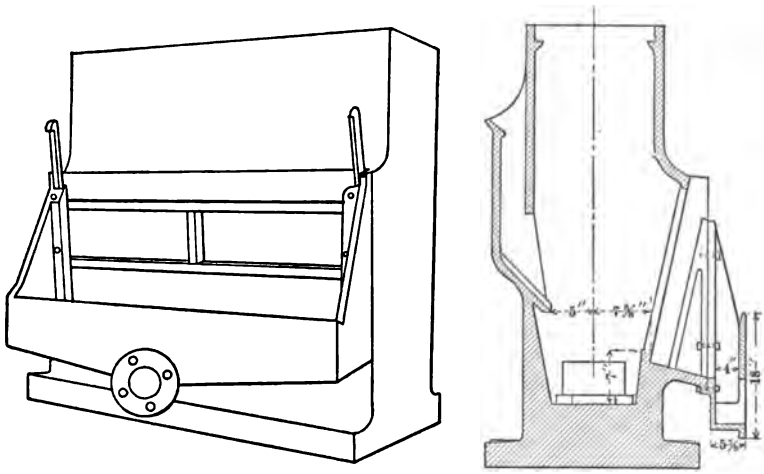
The cyanide solution is generally introduced into the battery through two 1.5-in. pipes which enter at the front of the battery between the first and second, and the third and fourth stamps. The flow of the solution through each pipe is controlled by an iron cock. In two cases a special mortar is used having a cast-iron collecting-launder bolted on at the front, which discharges centrally into the main launder, the latter collecting the pulp from all the batteries. A view of this mortar is shown in Fig. 1, which gives also the form and dimensions

of the mortar used at the Maitland mill. Generally all the screens are overhung with heavy canvas so as to avoid splash.

The Separation of the Sands from the Slimes by means of Cone-Classifiers.

The separation of the sands from the slimes is now done almost entirely by means of the outer sheet-iron cones of the common hydraulic classifier, the inner cones having been removed. At the outset, the problem of removing the sands from the slimes, when the ore is crushed in cyanide solution together with considerable lime, is more difficult than when crushed in water

Fig. 1.



PERSPECTIVE OUTLINE CROSS SECTION

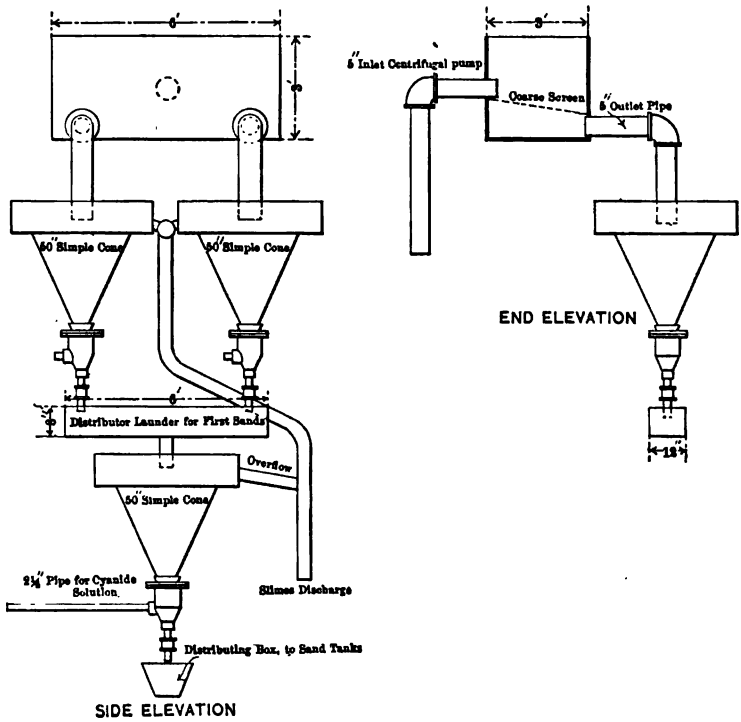
Mortar in Use at the Maitland Mill, So. Dak.

with practically no lime. The lime gives much trouble in two ways—first, by its coagulating-effect on the slimes which causes them to settle with the sands and coat the sand-particles with slimes, and second, by the formation of an excessive quantity of froth or foam, which, in itself alone, is a great nuisance about a mill. Fig. 2 shows the general arrangement of the classifying-cones.

The pulp is discharged from the battery by a launder leading to a central sump, from which it is raised to the cones by a pump. At three mills a Frenier spiral sand-pump is used for this purpose and at one a centrifugal pump. At the Lund-

berg, Dorr & Wilson mill a bucket-elevator is used. On account of lesser wear the Frenier pump is preferred for raising the battery-pulp, consisting of sands and slimes, but for the transference and agitation of slimes a centrifugal pump is generally used. The common size of the spiral sand-pump used for this work is 54 in. by 10 in., which will readily handle from 350 to 450 tons of pulp per day. They are operated at 19 or 20 rev. per min. The pulp is raised from 15 to 20 ft., the latter height

FIG. 2.



GENERAL ARRANGEMENT OF CLASSIFYING-CONES.

being the maximum practical lift for the diameter of pump stated above. For greater lifts a tandem arrangement of pumps is made.

The discharge of the sand-pumps being intermittent, a distributing-box is used at all of the mills in order to steady the outflow and give a uniform feed to the cones. These distributing-boxes are of different forms at the various mills. At the Horseshoe mill, a pyramidal box is used, 4 ft. by 4 ft. in cross-section at the top, having sides that slope 60° to meet at a

point. The inverted pyramid is topped by a box 12 in. high, through which enter the two 4-in. pipes from the sand-pumps. About 12 in. from the bottom of the pyramid, four 3-in. pipes emerge, one at each side, which feed into four 50-in. cones. The distributor is placed centrally over the four cones, and as low as possible so that the head, under which the discharge takes place, will be small. A screen placed in the distributor-box serves to keep foreign matter from passing to the cones.

At the Maitland mill, a plate-steel box, 6 ft. long, 3 ft. wide, and 3 ft. in depth, is used as a distributor. On one side, 21 in. from the top, two 5-in. pipes enter from the sand-pumps and discharge upon an inclined screen. The two 5-in. discharge-pipes, which feed the two 50-in. cones, have their centers placed 4 in. above the bottom of the distributor-box. Similar to the arrangement at the Horseshoe mill, the distributor is set as closely as possible to the cones. At the Dakota mill a similar box made of wood is used. The upper cones are simple cones of sheet-iron, from 40 in. to 50 in. in diameter, having vertical 12-in. sides at the top. The cones slope 60° and end in a 6-in. sorting-column, which has a 2-in. discharge controlled by an iron cock. The charging-pipe feeds at the center of the cone just below the level of the pulp. In most cases the top cones are closely covered by either a wooden or an iron cover, in order to confine the foam,—an arrangement which has the slight disadvantage of preventing a ready inspection of the cones. The upper cones are simple settling-cones.

The pulp passing to the cones contains from 14 to 19 per cent. of solids, of which from 30 to 50 per cent. are slimes and the rest sands. It is difficult to define exactly the terms "sands" and "slimes," but it is generally accepted by the men in charge of the plants crushing siliceous ores, that material finer than 200-mesh size constitutes a slime, while coarser material is a sand. The term slime has also been defined as that portion of the crushed ore which will render water muddy when added to it, while sands, no matter how fine, settle practically at once and do not remain in suspension.[4]

[4] What Constitutes a Slime? by W. J. Sharwood, *Engineering and Mining Journal*, vol. 76, pp. 538, 650. This definition is given by Mr. John Gross, in a paper, Cyanide Practice at the Maitland Properties, published by the Black Hills Mining Men's Association, 1904.

The overflow from the upper cones, containing practically no sands, passes to the slime-tanks by the overflow-launder. The sands which are discharged at the bottom of the cones, containing from 20 to 35 per cent. of slimes, are distributed by a short box to the lower cones, which are similar in construction to the upper ones, but have introduced into the sorting-column an upward current of cyanide solution either battery-solution or barren solution (solution that has been precipitated), generally the former. This solution is introduced through a 2-in. pipe having a cock so as to regulate the flow. The quantity of solution introduced in this way for a 42- to 50-in. cone, aggregates from 60 to 80 tons per 24 hours. The number of lower cones is always one-half that of the upper cones. The solution-pipes, entering the sorting-column of the lower cones, do not come directly from the stock-tanks, but from a special box provided with an overflow at a definite height, so that the head of the entering solution is always constant.

The final sand-discharge, consisting of from 25 to 30 per cent. of solids, and containing from 1 to 5 per cent. of slimes, passes to the Butters's distributors over the sand-vats, battery-solution being added in the carrying-launder so as to have 5 parts of solution to 1 part of sand. The mills try to make a close separation of sands from slimes in order to get a good leaching-rate in the sand-vats, said rate usually ranging from 2.5 to 3.5 in. per hour, although at one plant it is only from 1.5 to 1.75 in. per hour. A close separation is also desirable in order to avoid the trouble caused by the settling of sands to the bottom of the slime-vats, where they remain practically unacted upon during the greater part of the treatment.

A very close and satisfactory separation, however, is not possible for two reasons,—first, on account of the inherent defects of the cones used as classifiers, and, second, because of the bad effect of the lime in causing the slimes to remain with the sands. For these reasons the classification adopted is to make a clean sand rather than a clean slime, this being the lesser of two evils. For example, at the Maitland mill, where the sands carry from 1 to 2 per cent. of slimes, the presence of 5 per cent. of slimes would give an unsatisfactory leaching-rate to the sands. In making sands of this kind, the slimes contain from 15 to 20 per cent. of fine sands, of which but a small portion

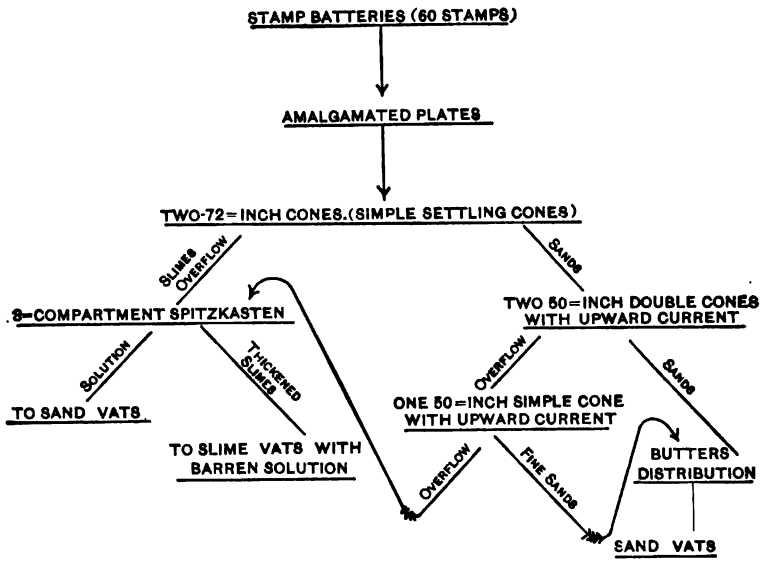
would remain on a 150-mesh screen. Of the ore crushed, the proportion treated as sands and as slimes varies at the different mills. At the Maitland mill, the average for 8 months was 48.2 per cent. as sands and 51.8 per cent. as slimes. At the Dakota mill the sands amount to from 65 to 70 per cent. and the slimes to from 30 to 35 per cent. At the Lundberg, Dorr & Wilson mill, the proportions of sands and slimes were approximately equal. At the Horseshoe mill, the slimes amount to from 26 to 30 per cent. and the sands to from 70 to 74 per cent.

Several different systems of classification by cones were tried before the one described was adopted. It should be noted that the system now in use re-classifies the sands from the upper cones, while, formerly, the plan was to reclassify, in the lower cones, the slime-overflow from the upper cones; but this practice was soon discarded because, in some instances, it gave unleachable sands. Double cones (the regulation cone-classifier) were also used, but most of the mills now classify with the outer cone only. The Hidden Fortune mill is the only one using a double cone with an upward current to re-classify the sands.

The Lundberg, Dorr & Wilson and the Hidden Fortune mills unwater the slimes before they are passed to the slime-vats; the former by means of a large sheet-iron cone, 22 ft. in diameter, the top portion sloping 40° and the lower portion near the discharge, 60°; and the latter by means of a 3-compartment spitzkasten 40 ft. long, 6 ft. wide and 8 ft. deep. In the spitzkasten the compartments are charged successively, and the thickened slimes drawn off and mixed with solution in the launder that transfers them to the slime-vats. The object of unwatering the slimes in this way is to give them an additional treatment with barren solution. If the slimes are not unwatered they pass to the slime-vat with battery-solution and are settled there for the first time; but with the unwatering-device above mentioned the slimes pass to the slime-vats with barren solution and have had one dilution by the time they reach the first slime-vat.

The scheme of the classification used at the Hidden Fortune mill, which is somewhat different from the practice at the other mills, is given in Fig. 3.

FIG. 3.



SCHEME OF CLASSIFICATION AT THE HIDDEN FORTUNE MILL.

Tables V. and VI. show the nature of the classification at the Dakota and the Lundberg, Dorr & Wilson mills.

TABLE V.—*Mechanical Analyses of Sands and Slimes at the Dakota Mill.*

Sands Constituting 70 Per Cent. of the Mill-Product.		Slimes Constituting 30 Per Cent. of the Mill-Product.	
Remaining	Per Cent.	Remaining	Per Cent.
On a 20-mesh screen.....	13 to 20	On a 100-mesh screen.....	0.3 to 0.4
On a 40-mesh screen.....	30	On a 150-mesh screen.....	12 to 33
On an 80-mesh screen.....	26 to 54	Passed a 150-mesh screen....	66 to 87
On a 100-mesh screen.....	7 to 8		
On a 150-mesh screen.....	13 to 18		
Passed a 150-mesh screen....	4 to 5		

TABLE VI.—*Mechanical Analyses of Sands and Slimes at the Lundberg, Dorr & Wilson Mill.*

Sands Constituting 50 Per Cent. of the Mill-Product.		Slimes Constituting 50 Per Cent. of the Mill-Product.	
Remaining	Per Cent.	Remaining	Per Cent.
On a 40-mesh screen.....	30	On a 60-mesh screen.....	0.5
On a 100-mesh screen.....	40	On a 100-mesh screen.....	1.5
On a 200-mesh screen.....	24	On a 200-mesh screen.....	18
Passed a 200-mesh screen...	6	Passed a 200-mesh screen....	80

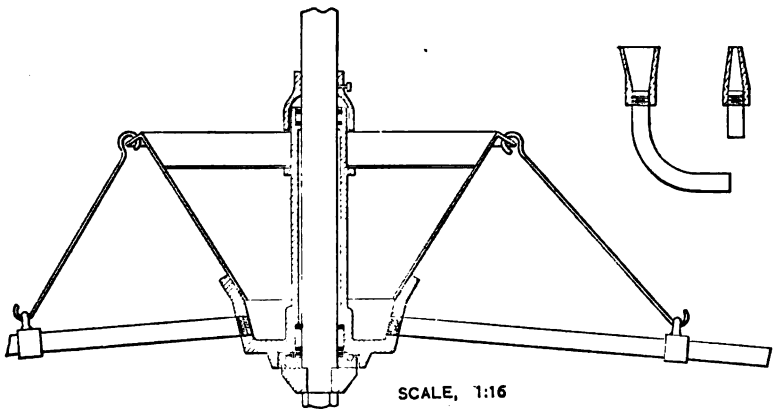
It is not possible to make a satisfactory comparison of mill-products at the different mills because the physical character of the ores treated by them differ to such an extent that what is considered a fine sand at one mill according to mesh-size, might be a slime at another mill.

The proper separation of the sands from the slimes is still a vital question for the plants of the Black Hills, and one that has given the mill men much trouble. The present system is a decided advance from the former practice, yet there is still much room for improvement.

The Treatment of the Sands.

The filling of the sand-vats is accomplished by Butters and Mein distributors, the construction of which is shown in Fig. 4.

Fig. 4.



Butters and Meins Sand-Distributor.

The distributor, suspended by a rod from a trolley running on tracks above the sand-vats, can readily be transferred from one vat to another. A launder feeds the pulp into the hopper of the distributor as near the center as possible in order to avoid throwing the feed against the sides of the hopper, which would cause an irregular distribution of the sands in the vat. The distributors vary in size according to the capacity required. The slope of the pipe-arms is 1 in 12, and the diameter of the pipes in the different distributors varies from 1.5 to 2.5 in. Generally all the pipe-arms in a distributor are of the same diameter, but at the Horseshoe mill the long arms are 3.5 in., the

medium arms from 2.5 to 3 in., and the short arms 2 in. in diameter. The discharge-nozzles, usually separate castings, are controlled by wooden plugs. Generally each distributor has six arms, although the one at the Horseshoe mill has eight. In one six-arm distributor the lengths of the arms are respectively 13.25 ft., 11.5 ft., 9.5 ft., 8.0 ft., 5.5 ft., and 2.5 ft. These arms are hung unsymmetrically in such a way that their weights balance the distributor. The discharge from the pipe-arms must cover the surface of the vat. The hoppers of the distributors are provided with a horizontal screen which keeps foreign matter out of the pipe-arms. The function of the distributors in the mills, crushing siliceous ore, is not, in part, that of a classifier acting with a filled tank in removing slimes from sands, but solely to distribute the pulp evenly in the vats. The sands are not laid down under water or solution, since the vat is "dry filled," and the incoming solution continually drained off through the filter until the vat is full of sands. The top layer of sands in the vat is always practically dry. The "dry-filling" method has two advantages,—(1) The slimes in the pulp are uniformly distributed with the sands in the vat (which is not the case when the material is filled directly under water), and for this reason the charge is more percolable; and (2) During the filling a large quantity of solution passes through the sands, and, in this way, the solvent action continues while the vat is being filled. Moreover, a charge laid down in this way is more porous than one laid down under water.

At the Maitland mill the quantity of solution which passes through a 140-ton charge of sands during the filling of the vat is 700 tons, which is equivalent to 5 tons of solution per ton of sands charged.

The time of filling a 30- by 6-ft. vat at the Maitland mill is 60 hours. At the Lundberg, Dorr & Wilson mill, an 18- by 10-ft. vat is filled in from 60 to 72 hours. At the Dakota mill a 115-ton vat is filled in 38 hours.

The indirect method of filling was formerly used at the first wet-crushing plants in the Black Hills, the Portland mill and the Dakota mill. The settling-boxes had two compartments which alternately discharged their contents into the sand-vats below, where the charges were raked and leveled. A double

treatment of the sands was first made at the Dakota mill, but was subsequently discarded as unnecessary after trial for a year. The settling-boxes soon proved so inefficient in classifying the material that they were replaced by the cone-system; and at some of the mills the sands were charged by distributors into the sand-vats filled with solution. All of the plants, however, soon adopted the more satisfactory method of "dry-filling."

The general method of treating the sands is the same at all the mills, although the quantity of solutions used and the time of treatment vary. The treatment of the sands is determined, as far as extraction will permit, by the handling of the mill-solutions, which, in a plant of the type under discussion, is quite complex. A few details of the sands-treatment are given in Table VII.

TABLE VII.—*Details of Sands-Treatment.*

Name of Mill.	Capacity of Vat.	Quantity of Solution Passing while Filling.	Quantity of Battery-Solution.	Quantity of Barren Solution.	Quantity of Wash-Water.	Total Time of Treatment.
	Tons.	Tons.	Tons.	Tons.	Tons.	Days.
Maitland.....	140	700	900	450	15	16
Dakota.....	115	.....	86	63	20	5
Horseshoe...	320	500	.....	.....	.....	8

The solutions leach through the sands continually, there being no "contact," or solution, standing on the ore, as is generally the case in dry-crushing practice. Furthermore, there is no strong solution, properly so called, although in some mills the battery-solution is slightly stronger than the barren solution, while in others the barren solution is slightly the stronger. At the Maitland mill, the battery-solution has from 1.2 to 1.3 lb. of KCN per ton, and the barren solution from 1.5 to 1.6 lb. of KCN per ton. At the Horseshoe mill, the battery-solution has 1.4 lb. of KCN per ton and the barren solution is somewhat stronger, though of undefined strength. At the Horseshoe mill the overflow-solution from the slime-vats, while they are being filled, is standardized in a sump-tank up to from 3 to 4 lb. of KCN per ton, and then passed through the sands. At the Dakota mill, the battery-solution contains 2.2 lb. of KCN per ton and the barren solution 2 lb. of KCN per ton. At the Lundberg, Dorr & Wilson mill, the battery-solution contains

2 lb. of KCN per ton, and at the Hidden Fortune mill it contains 1.3 lb. of KCN per ton.

The quantity of wash-water used varies but little at the different mills, amounting to from 0.1 to 0.2 tons per ton of sands treated. A small quantity only of wash-water is required, since in most of the mills all the cyanide solutions are weak, and very large quantities of solution are used. The deficit of solutions in the mills is made up mainly from the wash-water obtained in treating the slimes.

The following data show the results on sands obtained during 5.5 months at the Dakota mill. The value of the ore averaged $4.75 per ton, while the value of the sand-tailings averaged $1.22 per ton, thus giving an extraction of 74.25 per cent. from the sands. The moisture, which passed out with the sand-tailings, had a value of $0.40 per ton. During May, 1904, the average value of the ore was $4.55 per ton, the average value of the sand-heads charged into the vats was $2.60 per ton, and the average value of the sand-tails, unwashed, was $1.06 per ton, which gives an extraction of 76.7 per cent. from the sands. A comparison of the original value of the ore, with that of the sand-heads and the sand-tails, shows a 42.8-per cent. extraction in the batteries and cones, and a 33.9-per cent. extraction during the sand-treatment proper.

At the Hidden Fortune mill, the extraction on the sands averages 75 per cent. The extraction from the slimes, as well as the total extraction, is given under the following section of this paper.

The Treatment of the Slimes by Agitation and Decantation.

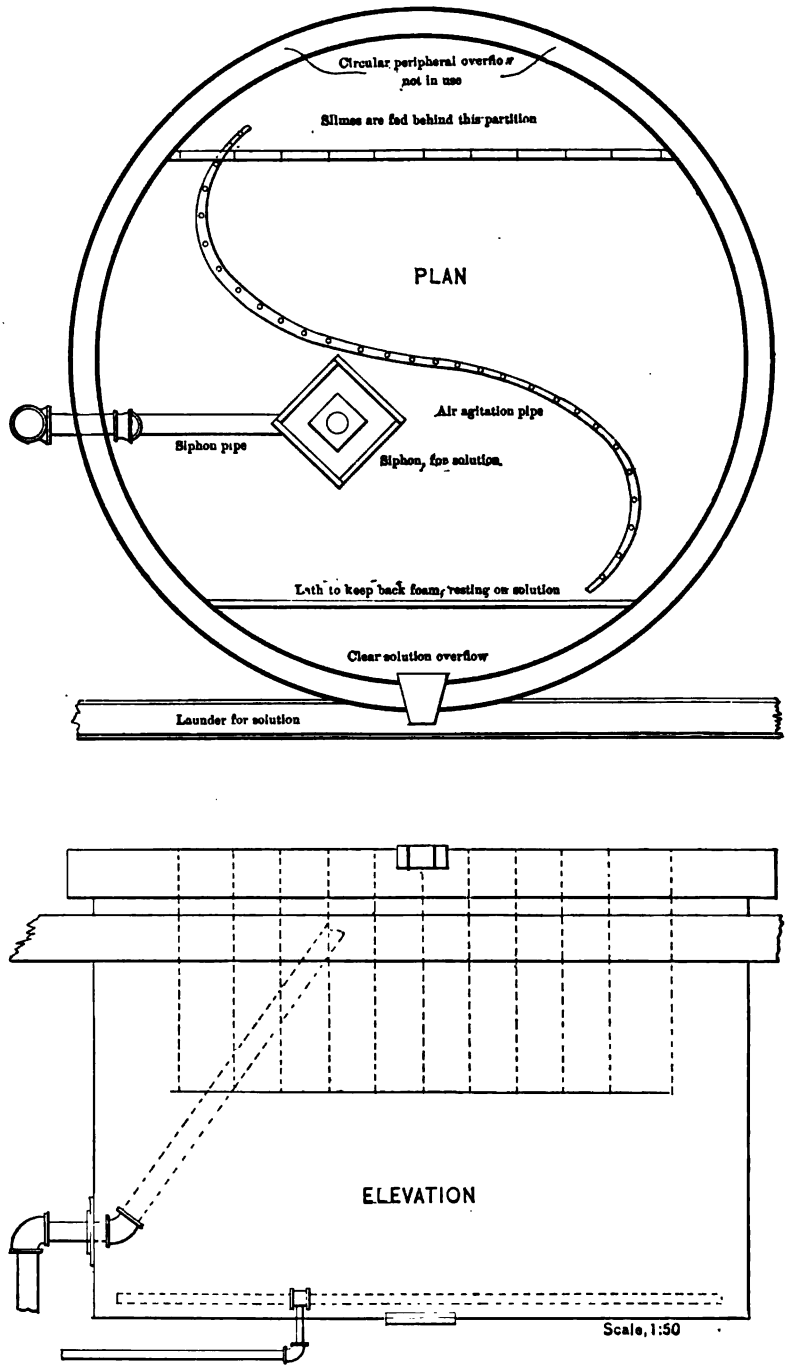
Two methods of slimes-treatment are practiced in the Black Hills; the first having the treatment of the slimes completed in the vat into which they are originally charged, and in which most of the agitation is done by means of compressed air; and the second, repeatedly transferring the slimes from one vat to another (generally 3 or 4 times) before the slimes are discharged, the agitation being done by means of centrifugal pumps.

The first method is practiced at the Horseshoe mill as follows:—There are 16 slime-vats, 14 ft. in diameter and 10 ft. deep, and two 30 ft. in diameter and 16 ft. deep. The details

of the construction of a slime-vat is shown in Fig. 5. A partition extends from the top nearly to the bottom, at one side of the vat, behind which the slimes are charged as they come from the cones. The vat is filled with barren solution, and the slimes are then charged; a strip of wood or lath resting on the surface of the solution holds back any foam and allows the surplus solution to pass off clear at the lip of the vat.

The slimes settle rapidly and the solution usually runs off clear at the lip until the slimes have accumulated to a depth of about 50 in., which is equivalent to from 25 to 30 tons of dry slimes. As soon as the solution at the lip becomes cloudy, the slimes-charge is turned into the next vat, and the slimes in the vat just filled are permitted to settle. This settling occupies about 10 hours, during which the supernatent solution is decanted by means of a simple wooden frame having a pipe at the center connected with a take-off pipe about 18 in. from the bottom of the vat. (See Fig. 5.) It is important to have the slimes settle as far as possible, and to decant as closely as possible, without taking off muddy solution, usually within an inch of the settled slimes. The main object of the slimes-treatment is to remove the dissolved values by successive dilutions, hence, unless the decantation is extremely close each time, this object will not be fully gained. As soon as the decantation is complete, about 40 tons of barren solution is added to the vat; the contents during the addition being agitated by the introduction of compressed air at 40 lb. per sq. in. The air enters the bottom through two S-shaped pipes (Fig. 5 shows one only), crossing each other, having 0.12-in. perforations. This air-agitation is the only one that the slimes receive, but it has the weak point that it fails to move all of the material (especially the heavier portion of the slimes and the fine sands) at the bottom of the vat. For this reason the contents of a vat are not sluiced out, but that portion of the slimes which will run out by the bottom gates is allowed to flow away, leaving from 2 to 4 tons of heavy thick slimes which form a portion of the succeeding charge, thus getting two treatments for this material. Each charge of slimes is washed four or five times with barren solution and once with water, the quantity of each wash amounting to 40 tons, which is equiv-

FIG. 5.



DETAILS OF CONSTRUCTION OF SLIME-VAT.

alent to 6.5 tons of barren solution and from 1 to 1.6 tons of wash-water, for each ton of dry slimes treated.

The discharged slimes contain approximately 50 per cent. of moisture. From ore averaging from $8 to $9 per ton in value, the dried slime-tailings averaged $1.75 per ton in value, of which from 30 to 40 per cent. still existed as gold in solution. The washed slime-tailings, therefore, carried $1.24, and the solution discharged as moisture, $0.52 per ton.

In the treatment of slimes by successive additions of solution for the extraction of the values, when in the total treatment, a definite quantity of solution per ton of dry slimes is used; it is theoretically necessary, in order to get the maximum extraction, to use this quantity in a comparatively large number of small additions, rather than in a few large ones. Thus, in treating a ton of slimes with 6 tons of solution, it is theoretically better to give six additions of 1 ton each, rather than two additions of 3 tons each; but in case any one addition should be larger than the others, it would naturally be applied to those slimes of highest value. It must be borne in mind, however, that during the slimes-treatment, values are constantly going into solution, so that while the values in the solution are being reduced by dilution, they are constantly being augmented by the solution of new values; for this reason, the solution, which is finally discharged as moisture with the slime-tails, will never be as low in value as the dilution requires. Since the cost of adding a definite quantity of solution to the slimes is the same, irrespective of the values extracted by these solutions, it is evident that the economic limit is soon reached, beyond which a further dilution will not pay. Few of the mills can afford to apply more than four or five dilutions. Moreover, the greater the number of dilutions, the larger the quantity of solution to be handled in the mill. Although, at the present time, the mills handle a large quantity of solution per day, the application of an extra water-wash to the slimes would not be very objectionable, provided the decantation from the last wash were to run to waste through a large zinc-box, containing the poorer grade of shavings and the dust from the lathe. The saving resulting from the increased extraction and the utilization of otherwise waste material would probably be appreciable.

The treatment of the slimes at the Hidden Fortune mill dif-

fers only in a few details from the practice at the Horseshoe mill. The slime-vats are first filled with barren solution and the slime-pulp is charged at the center of the vat through a large pipe extending to within a few feet of the bottom.

The clear solution overflows continually around the entire periphery and is collected by an annular launder and taken to the battery-solution sumps. When the outflowing solution becomes cloudy, the charging of the slimes is stopped and the slimes are allowed to settle; meanwhile, the clear solution at the top is decanted to the battery-solution sump, similar to the practice at the Horseshoe mill. When the solution has been decanted to within an inch or two of the slimes, the top layer of thin slimes to a depth of 3 or 4 in. is pumped to the vat that is being filled with slimes, with the object of removing as much solution as possible, in order to have a maximum efficiency from the next dilution. The first wash of barren solution is then added through the perforated air-pipes at the bottom of the vat, which not only keeps the perforations clear, but effects, as well, the agitation and moving of the heavier slimes at the bottom of the vat. As soon as the addition of the barren solution is ample, the charge of slimes is agitated by introducing air at a pressure of 40 lb. per sq. in., and then allowed to settle, the clear solution being decanted, as already described above. The slimes are washed three times with barren solution and once with water, but before they are finally discharged from the vat, the top layer to a depth of 3 or 4 in. is again pumped to the vat which is being filled. The total time required for the treatment of the slimes is from 3 to 4 days. At the Hidden Fortune mill, three washes are found to be sufficient, since the slimes are unwatered before they pass to the slime-vats. At this mill, the extraction from slimes, determined by assaying the washed tailings, is 80 per cent., but the actual recovery is but 75 per cent., determined by assaying the unwashed tailings; these results show a 5-per cent. loss of gold which passes out in solution with the slime-tailings.

The second method of treatment, in which the slimes are successively transferred from one vat to another, is illustrated by the practice at the Dakota mill. Here the slimes from the cones are alternately charged into the loading-vats Nos. 1 and 2, which are 20 ft. in diameter and 137 in. deep. Each vat is

filled for 12 hours and then allowed to settle for 10 hours. The clear solution at the top is continually decanted off and finally withdrawn to within 1 in. of the settled slimes, which are then pumped by a 4-in. suction centrifugal pump to vat No. 3, barren solution being continually added to the suction of the pump during the transferral, which occupies from 1 to 2 hours. In vat No. 3, the slimes are permitted to settle again for 10 hours, the clear supernatent solution being decanted off meanwhile. The slimes are then pumped to vat No. 4 (barren solution being added to the suction as before), and receive an additional agitation for about an hour by pumping the slimes from the bottom and returning them over the top of the vat. The settling and decanting is then repeated, and the slimes transferred to vat No. 5, from which, after settling and decantation of solution, they are transferred to vat No. 6; wash-water is used this time instead of barren solution. The quantity of wash-water added is equivalent to the quantity of moisture in the slimes. The last settling takes place in vat No. 6, the solution is decanted and the slimes are discharged. This total time required for the treatment of the slimes is 5 days.

In order to coagulate the slimes, from 4 to 6 lb. of lime per ton of ore are added at the battery. This addition of lime, being the only one made in the mill, must be carefully adjusted; if too little be added, the slimes will not coagulate readily, and if too much, there will be subsequent trouble in the precipitation of the values.

The following data of slimes-treatment at the Dakota mill are of considerable interest. During 5.5 months 6,681.67 tons of slimes were treated, which amounted to 33 per cent. of the total quantity of ores crushed. The average value of the ore during this period was $4.75 per ton. The slime-tailings, dried and unwashed, assayed $1.31 per ton, thus giving a recovery of values from the slimes of 72.28 per cent. The washed slime-tailings assayed $0.912 per ton, showing a solution of the values of 80.7 per cent., and a loss of soluble gold of $0.40 per ton, or 8.42 per cent. of the value of the ore. The slimes are discharged with 50 per cent. of moisture, which consists of solution having a strength of 1.07 lb. of KCN per ton. Since there is a ton of this solution going to waste for every ton of dry slimes discharged, there were lost during the above period

7,147.6 lb. of KCN, which, at $0.23 per pound, has a value of $1,644. Adding to this, the loss in dissolved gold, amounting to $2,672, the total loss was $4,316, or $0.64 per ton. These figures show clearly the weak points of the decantation-system of slimes-treatment. The Dakota mill is one of the most successful mills in the Black Hills, treating practically the lowest grade of siliceous ores handled in the district.

The results of the sands-treatment at this mill have already been discussed earlier in this paper.

The treatment of the slimes at the Maitland mill is similar to that at the Dakota. To each ton of dry slimes is added 6.38 tons of barren solution and 0.96 tons of wash-water. The solution, passing out with the slimes as moisture, contains gold to the value of $0.46 per ton. The head-slimes solution, or the solution first decanted from the slimes while filling, has a value of about $2 per ton.

At the Lundberg, Dorr & Wilson mill, the Moore slimes-process is used for the slimes. In this process there are 3 rectangular tanks, 15 ft. long, 7 ft. wide, and 5.5 ft. deep; the first has a double-hopper bottom, the sides inclined 45° in order to aid in the circulation of the pulp and to discharge more readily some heavy slimes which may fail to adhere to the filter-frames and fall to the bottom of the vat. There is a set of 34 frames, each 4.5 ft. by 6 ft. in size, made of 2-in. material. The filter is made of 18-oz. duck. Both sides of the frames are effective as filters, and the total filtering-area is therefore 1,836 sq. ft. The interior of the frames is connected with a pump and a compressor, which furnish suction or pressure respectively. The suction is equivalent to 18 in. of mercury. The set of frames is suspended from a hydraulic crane which transfers it from one vat to the other.

The method of treatment is as follows:—The slimes, after agitation by a centrifugal pump in an 8-ft. sheet-iron cone, are passed to the first vat, the frames are immersed in the slimes and the suction is started. A coating of slimes deposits on the filters, and the clear solution is discharged by the pump. At the expiration of from 40 to 65 min., the layer of slimes on the filters has accumulated to the thickness of an inch, which is equivalent to a load of 4 tons, the set of frames is then lifted out with its adherent load of slimes (suction meanwhile being

continued), and immersed in the next vat filled with barren solution, which is sucked through the slimes for 40 min. The frames are then transferred to the next vat, which is filled with water. In like manner the wash-water is sucked through the slimes for 40 min., and the frames are then lifted out and carried to a position above the discharge-hopper. The suction is changed to pressure, which causes the slimes to peel off into cars placed below the hopper. A little scraping has to be done in order to clean the frames.

There being no data yet available concerning the working-results of the Moore process, a comparison with the decantation-process cannot be made. The Moore process, however, discharges dryer slimes; those at the Lundberg, Dorr & Wilson mill containing from 34 to 36 per cent. moisture, as compared with 50 per cent. of moisture, which is the usual proportion of the slimes from the decantation-process.

At the mills where the upper layer of slimes is pumped off, as described for the Hidden Fortune mill, the slimes are discharged with from 46 to 47 per cent. of moisture. At some of the mills of the district, where the tailings must be confined within narrow limits and hence cannot flow freely to waste, considerable trouble is experienced from the high content of moisture in the slime-tailings. The slime-tailings from the Moore process are much more easily held in check.

In making a comparison between the Moore process and the decantation-process, it must be borne in mind that the slimes in present practice are under treatment in the former only from 2 to 3 hours, and in that time they receive but little agitation, so that the solution of the gold must take place practically before the slimes go to the Moore process. However, the Moore process, even with separate agitation, shortens the time of the slimes-treatment materially, and, in addition, a plant of large capacity can be installed within a small area.

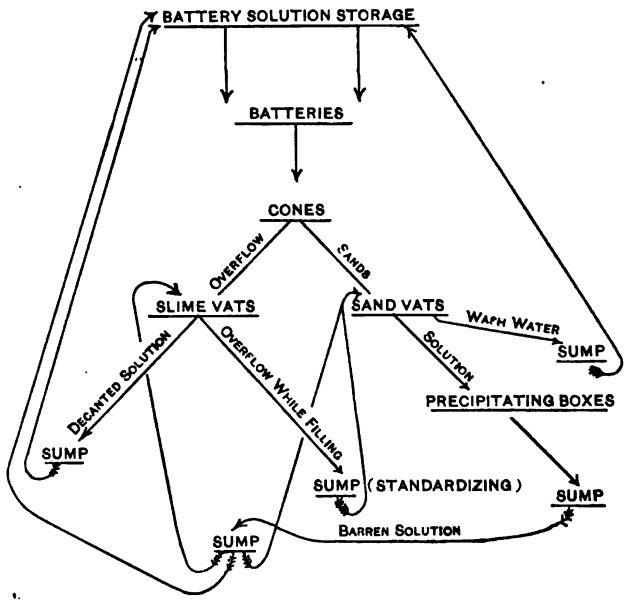
Filter-press experiments on a fair-sized scale have been made at one of the mills in the district which indicated that slimes could be made containing about 25 per cent. of moisture; and that these slimes, on account of close washing, carried very little cyanide and dissolved gold. It would not be surprising to see filter-pressing, at least in part, eventually replace the decantation-process in the Black Hills district.

The Distribution of Solutions in the Mills.

The general manner of distributing the solutions in the mills is quite complex, as is shown by Fig. 6.

In this diagram it appears that the only solution passing to the precipitation-boxes is that which has gone through the sands, amounting in quantity to from 800 to 1,000 tons per day. The battery-solution has a gold-value of approximately $1 per ton, which is derived, for the more part, from the decanted slimes-solution. The cyanide needed to strengthen the solution

Fig. 6.



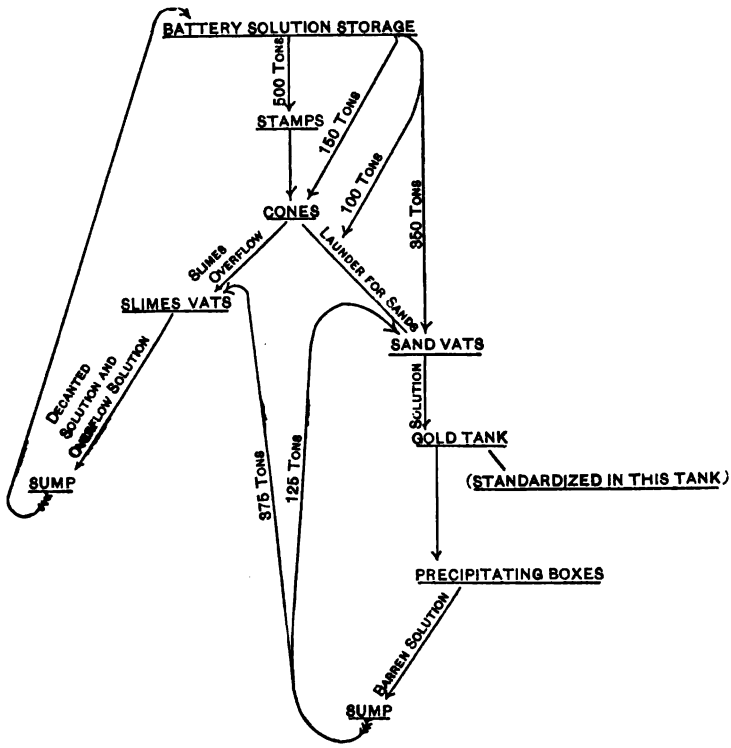
Distribution of Solutions at the Horseshoe Mill.

is added to a comparatively small quantity of decanted slime-solution, preferably the overflow-solution from the slime-vats while they are being filled. This solution is brought up to a strength of from 3 to 4 lb. of KCN per ton, and is run onto the sands, so that they get the benefit of what is practically a strong solution.

The scheme at the Maitland mill, given in Fig. 7, shows that the only solution precipitated is that which has passed the sands, a practice which is general at all of the mills. The solution, which contains some copper, is re-standardized ahead of the zinc-boxes, in order to get a more efficient precipitation.

The strength of the various solutions at the different mills is given under the discussion of the treatment of the sands. The battery-solution, at the Maitland mill, has a value of about $0.50 of gold per ton, and the barren solution about $0.10 per ton. Daily, about 1,100 tons of battery-solution are pumped, and nearly 500 tons of solution are precipitated, which, after precipitation, becomes the barren solution; so that the mill

FIG. 7.



DISTRIBUTION OF SOLUTIONS AT THE MAITLAND MILL.

handles about 1,600 tons of solution per day, which, at a capacity of 120 tons of ore per day, is equivalent to 13.1 tons of solution per ton of ore. The pumping-expense of a wet-crushing mill per day is appreciable; and in the design of a mill, the question of handling the solutions is a very important one. At one of the mills the re-standardization takes place in the battery-sump, so that the strongest solution is used in the battery. For obvious reasons this practice is not the best.

The Precipitation of Gold.

Generally, the precipitation-process as used at the present time, gives no trouble. In the earlier practice of passing the decanted slimes-solution through the zinc-boxes with the idea of keeping the battery-solution practically free from gold-values, trouble was experienced by getting a very bulky and low-grade precipitate, due to the accumulation of fine ore-slimes in the boxes. Precipitation is effected in barrels at the Dakota mill and Horseshoe mill. At the latter there are 120 barrels in use, each 2 ft. in diameter and 2 ft. high, holding 5 cu. ft. (25 lb.) of zinc. The barrels are readily lifted by a pulley-gear running on a trolley, and conveyed to the clean-up tank for discharging and cleaning. At the Horseshoe mill, treating from 225 to 250 tons of ore per day, 1,000 tons of solution are precipitated every 24 hours, at a zinc-consumption of 1 lb. per ton of ore treated. At the Dakota mill the zinc-consumption is 0.58 lb.

At the Maitland mill, the precipitation is effected in the usual form of iron zinc-boxes. At the former, treating 120 tons of ore per day, there are four 8-compartment boxes, each compartment having a capacity of 7 cu. ft. of zinc. From 350 to 490 tons of solution are precipitated per day, averaging 2.13 tons of solution for each cubic foot of zinc. The solution entering the boxes is kept at a strength of 2.5 lb. of KCN per ton, for the reason that a lower tenor causes trouble by the precipitation of the copper, which the solutions contain to a small extent. The zinc-consumption here is 0.3 lb. per ton of solution precipitated, or 1.33 lb. per ton of ore treated.

Treatment of the Precipitates.

Without exception, the precipitates are refined by the usual sulphuric-acid method, and the refined material is melted into bullion and sold to the United States Assay Office at Deadwood. The fineness of the bullion varies at the different mills, depending on the silver-content of the ores, which is of a wide range. However, at most of the mills, the bullion produced is from 450 to 600 fineness in gold. In order to sell the bullion to the Deadwood Assay Office, which will not buy any product of a fineness less than 600 in gold, some of the mills smelt the precipitates without the addition of metallic iron and hence with the formation of considerable silver-matte, so that the resultant

bullion will be above the minimum limit. The slags from the smelting of the precipitates are generally sold to the smelters at Denver, although, at the Horseshoe mill, the slags and matte from the meltings are refined by melting with litharge and cupelling the resultant bullion.

General Extraction Data.

The extraction at the mills varies from 65 to 75 per cent., according to the character of the ores treated. At the Maitland mill, where half of the ore is treated as slimes and half as sands, the extraction, amounting to 73.2 per cent., is distributed as follows,—39 per cent. in the batteries and cones, 16.5 per cent. in the slimes-treatment, and 17.7 per cent. in the sands-treatment. Generally, the recovery of bullion is either from 2 to 3 per cent. higher, or lower, than the results calculated from the assays of the ores, tailings and solutions.

The Cost of Treatment.

The detailed cost of treatment at the Dakota mill during part of the year 1902, while treating 100 tons of ore per day, is given in Table VIII. The average value of the ore treated at the mill was $4.70 per ton.

Table VIII.—*Cost of Treatment Per Ton at the Dakota Mill.*

Labor	$0.453	Repairs	$0.027
Superintendence	0.090	Refining	0.030
Cyanide	0.211	Assay office	0.040
Zinc	0.033	General expense	0.050
Lime	0.012		
Power	0.226	Total	$1.277
Shoes, dies, etc.	0.095		

The present cost of treatment at the Dakota mill, owing to a slightly increased capacity, has been reduced to $1.17 per ton.

The cost of treating about 125 tons of ore per day at the Maitland mill (not situated on the railroad, and threfore subject to considerably higher expense for supplies) is $1.79 per ton. The cost per ton at the other mills ranges from $1.17 to $1.79.

In conclusion, I wish to acknowledge my indebtedness, for much of the information contained in this paper, to Mr. John Gross of the Maitland mill, to Mr. John Ingersol of the Dakota mill, to Mr. J. V. N. Dorr of the Lundberg, Dorr & Wilson mill, to Mr. Freeman Steele of the Hidden Fortune mill, and to Mr. G. H. Clevenger of the Horseshoe mill.

Cyanide Practice at the Maitland Properties, South Dakota.

BY JOHN GROSS, MAITLAND, SO. DAK.

(Lake Superior Meeting, September, 1904.)

INTRODUCTORY.

THE group of claims, comprising over 1,100 acres, located at Maitland (formerly known as Garden City), in the Ida Gray mining district, Lawrence county, South Dakota, are now being developed and operated by Alexander Maitland, the owner.

Prior to the acquisition of this property two years ago by Mr. Maitland, but little work had been done in this district, and little was known of the extent of the ore-deposits. Recent development, however, has opened up some very promising ore-shoots.

The ores so far encountered are the so-called "Potsdam siliceous ores," occurring in the flat formation lying immediately above the Cambrian quartzite, which dips at this locality about 12°. Both oxidized and blue ores are found, and the latter have proven particularly refractory to treat. No prospecting in the underlying slates of the Archean period has so far been attempted; but this work is one of the future considerations in the development of the property.

Small quantities of blue ores are unavoidably sent to the mill, but special care is taken to send only the oxidized ores for the cyanide treatment. The ores, being close-grained and hard, have given considerable trouble in their treatment. Pyrrhotite and pyrite exist in about equal proportions; arsenic, copper, and traces of antimony and tellurium are encountered, and considerable quantities of bismuth have been found in the zinc-box precipitates. The silver in the low-grade ores is slightly greater than the gold, while in the high-grade ores the reverse is generally true.

Table I., showing the variations in the analyses of these ores, gives a fair idea of their general character.

Early in 1902, the building of a 40-stamp wet-crushing cyanide-mill was begun, and it was finally placed in commission

TABLE I.—*Analyses of Maitland Ores.*

	Oxidized Ore From General Mill-Samples.	Blue Ores.	High-Grade Ores, Both Oxidized and Blue, Mainly the Latter.
	Per Cent.	Per Cent.	Per Cent.
SiO_2	70.00 to 80.00	65.00 to 80.00	70.00 to 85.00
Fe	10.00 to 12.00	7.00 to 15.00	8.00 to 15.00
S	0.50 to 2.00	5.00 to 12.00	2.00 to 12.00
As	none to 0.30	0.50 to 2.00	trace to 0.30
Sb	trace	trace	trace
Te	trace	trace	trace
Cu	trace to 0.02	trace to 0.02	trace to 0.02
Zn	none	none	trace
Mn	trace to 1.00	trace to 0.50	trace to 1.00
Al_2O_3	2.00 to 5.00	2.00 to 6.00	1.00 to 2.00
CaO	2.00 to 4.00	1.50 to 3.00	0.50 to 1.00
MgO	0.50 to 2.00	trace to 0.50	trace to 0.50
	Ounces per Ton.	Ounces per Ton.	Ounces per Ton.
Au	0.30 to 0.60	0.50 to 0.90	1.00 to 3.00
Ag	0.50 to 2.00	1.00 to 6.00	0.50 to 2.00

January 1, 1903. The treatment, briefly, was to crush the ores in a cyanide solution (as practiced at several of the local mills, and originated and brought to a high degree of perfection at the large mills in New Zealand), to separate the sands and slimes, to leach the sands, and to treat the slimes by agitation and decantation.

The ore, trammed from the shaft, 330 ft. east of the mill, through a covered tramway, is delivered to the crusher-bin; hoisting being done only during the day-shift, six days a week, which gives sufficient supply of ore to keep the mill running steadily.

The boilers are near the hoist, and the steam is carried to the mill through a 6-in. covered pipe passing through the tramway.

Convenient to the mine and the mill, and connected therewith by a tram-track, are a large blacksmith shop containing two forges and a trip-hammer, and a well-equipped machine-shop having a 20-ft. lathe, a 31-in. drill-press, a planer and a pipe-cutting machine.

The melting-room is entirely apart from the mill, and is situated near the general office, thereby obviating the necessity of having any fires in the mill. The heating in the winter is done by a series of 8-in. galvanized-iron pipes, in all about 1,500 ft. long, which take the exhaust-steam from the mill-engine, and causes practically no back-pressure on the engine. Sufficient heat is obtained to keep the building at a comfortable

temperature; and during the coldest weather of last winter, with an outside temperature of 0° F., the temperature of the interior of the mill was from 54° to 62° F.

The water for the mill and boilers is obtained entirely from

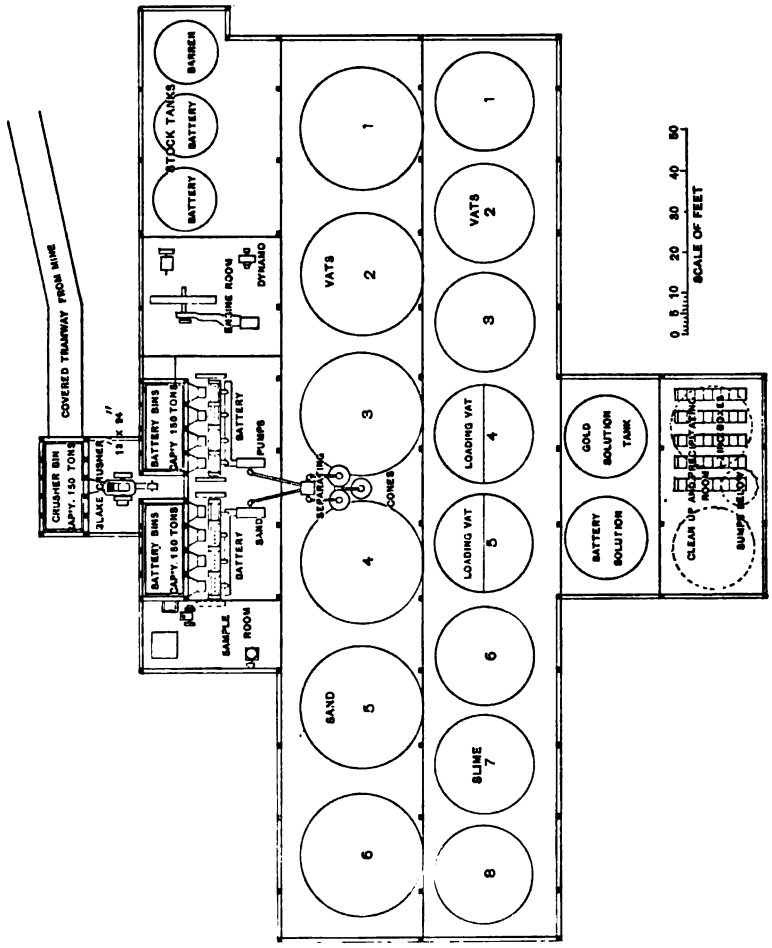


FIG. 1.—THE MAITLAND MILL, MAITLAND, SO. DAK. GROUND PLAN.

the mine workings, and is stored in a 50,000-gal. tank on the hill back of the mill.

MILL PRACTICE.

A ground plan of the mill-building is shown in Fig. 1 and a cross-section through the center of the mill in Fig. 2. A legend of the metallurgical treatment at the mill is given in Fig. 3.

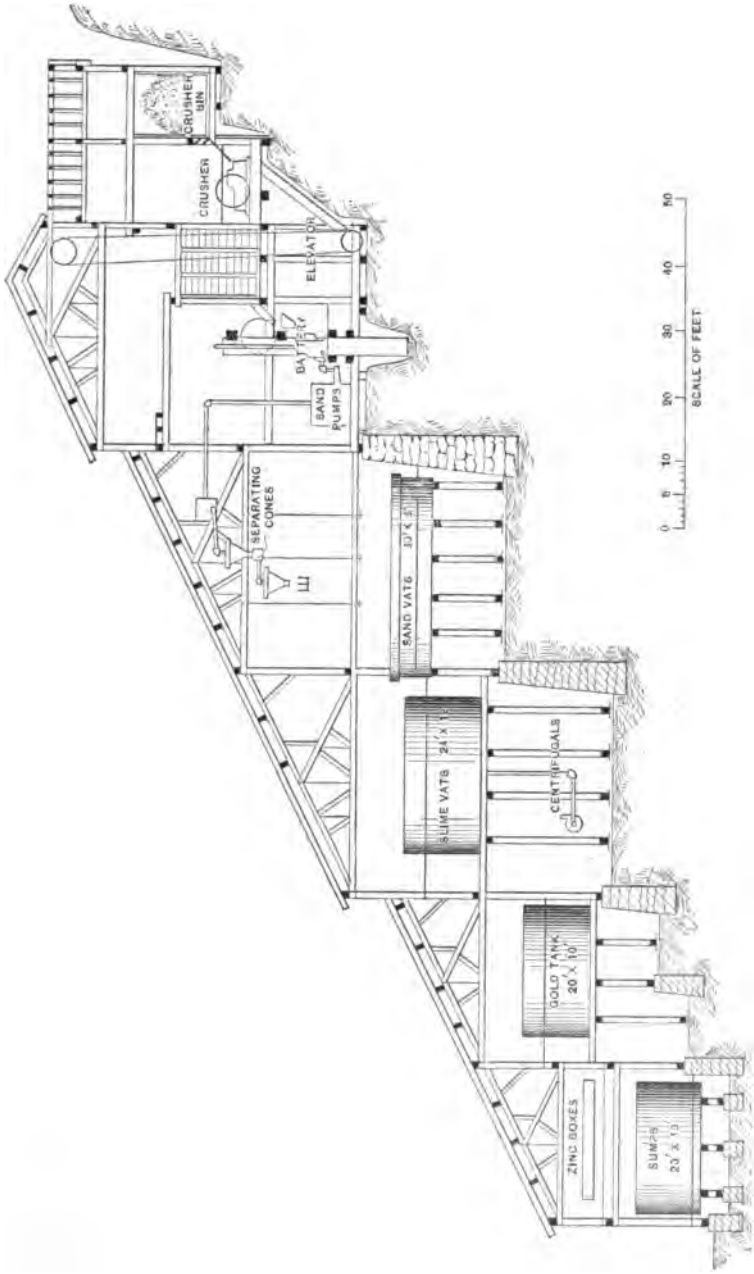


FIG. 2.—THE MAITLAND MILL, MAITLAND, SO. DAK. CENTRAL CROSS-SECTION.

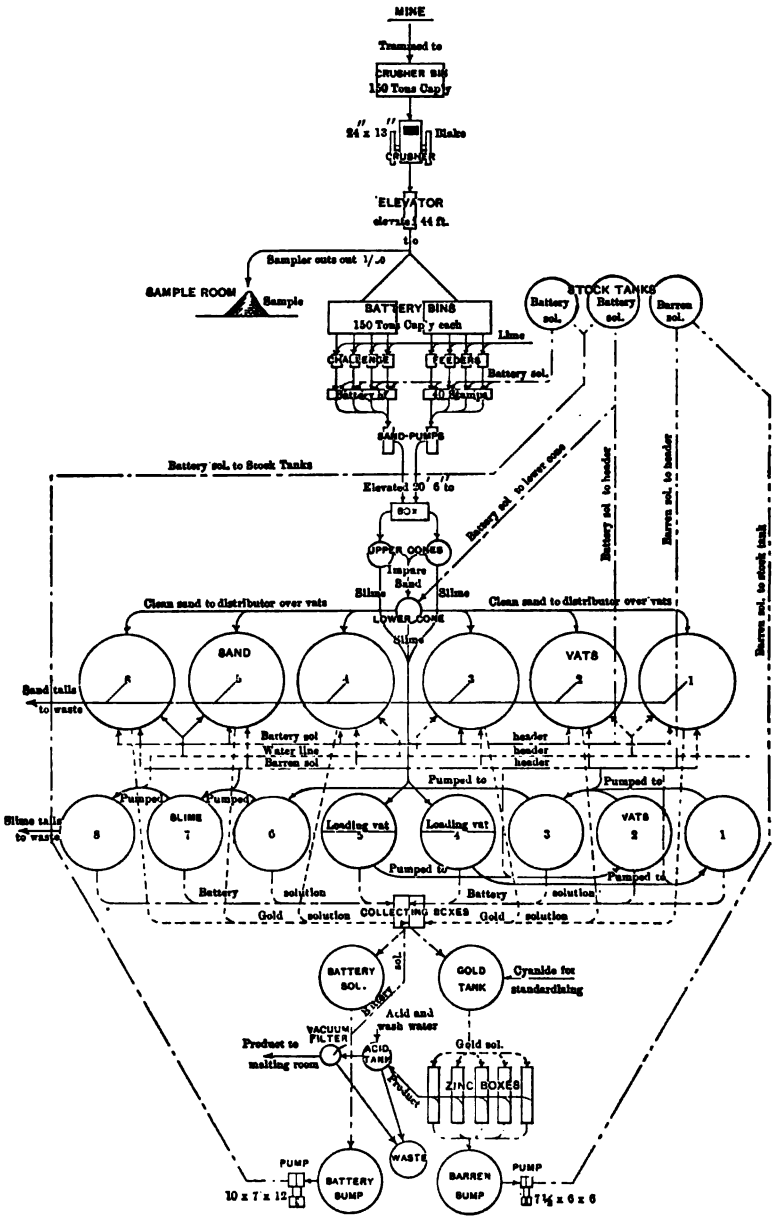


FIG. 3.—LEGEND OF OPERATION AT THE MAITLAND MILL, MAITLAND, SO. DAK.

The statistical data given in this paper are based on a daily tonnage of from 110 to 120 tons, which is the rate of treatment now maintained.

Crushing the Ore.—The ore as it comes from the mine, carrying an average of 8-per cent. of moisture, is delivered to the crusher-bin of 150 tons capacity. The crusher-bin has a flat bottom, and is 22 ft. long, 10 ft. wide and 12.5 ft. deep, having a capacity of 2,750 cu. ft.; the inside is lined with double planking, and is protected at the points most exposed to wear by 0.25-in. steel plates. Upon the assumption that the ore will form its own slope, the idea of using a rectangularly shaped bin, in place of one having a bottom sloping to the discharge, seems, at first glance, to be economical, but this form is not as satisfactory as the 45° sloping-bin, which practically discharges the ore cleanly. It is true that wet ore forms its own slope, but this slope is so steep that the working-capacity of the bin is reduced to a very low point of efficiency. The ore for the breaker is passed directly from the bin to the breaker, no grizzly being intermediate; it is contemplated, however, to install a grizzly, not only to shorten the actual running-time of the breaker, but as an aid in breaking very wet ores.

The breaker is a 24-in. by 13-in. Blake, running at 260 rev. per min., and set to break to a size that will pass through a from 1.5- to 2-in. ring. The average time of running is 7 hours per day, which gives a capacity of from 16 to 17 tons per hour. A set of jaw-plates in the breaker lasted 15 months, during which time 41,000 tons of ore were broken.

The crushed ore is passed through iron-lined chutes, inclined 45°, to the elevator-boot, and is elevated 44 ft. for delivery to the battery-bins. In the original mill a bucket-elevator of the continuous type, traveling 100 ft. per minute, was provided; but was soon discarded after handling 9,300 tons of ore, for the reason that the links wore out and caused it to fall into the pit on several occasions. A 14-in., 8-ply rubber belt is now used, traveling 350 ft. per minute; and, in order to lengthen its life, it is re-inforced between the buckets with old pieces of belt which take the roughest wear. One belt has so far been worn out after having handled 16,965 tons of ore at a cost of 0.91 cents per ton. The first rubber-belt elevator received extremely hard usage during the first few months of its service; the

second one, now in place, has handled a larger tonnage and is still in good condition.

The 12-in. buckets are spaced 18 in. apart on the belt. No. 10 sheet-steel buckets were first used, and handled 16,965 tons of ore at a cost of 0.45 cents per ton. Malleable-iron buckets proved too light for the work and handled 8,080 tons of ore at a cost of 0.76 cents per ton. No. 6 sheet-steel buckets are now used, and the tendency is to use still heavier ones.

The stream of ore, in falling from the elevator-head to the battery-bins, is cut by an automatic sampler, consisting of two light endless-link belts carrying several buckets suspended between them; the chains which travel at the rate of 100 ft. per minute cause one of the buckets to pass through the falling stream of ore every 30 seconds, cutting the full width of the stream and taking 2 per cent. of the ore as the sample which is delivered to the sample-room, where it is cut down and quartered in the usual manner. The sample-buckets are 5 in. wide, 14 in. across and 6 in. deep, and are ample for a capacity of 25 tons per hour. This sampling-arrangement has proven very satisfactory and reliable.

Stamps.—The two battery-bins, each of 150 tons capacity, are situated one behind each set of 20 stamps. The ore falls through chutes to suspended Challenge feeders, one for each battery of five stamps. The mortar is narrow, of single-discharge type, 12 in. wide at discharge, and a 6-in. discharge is maintained, crushing the ore to 26-mesh size. The screens used are 26- by 13-mesh, No. 26 wire, and, due to the longer opening, they do not clog so readily. These wire screens are also rolled, which helps to keep the holes open. The life of the screens now installed is about 50 days. Two sizes of screen-frames are used; the larger size taking a 12-in. clear height of screen, and, after it has become worn at the top and bottom, it is removed, cut down and placed on the smaller frame, taking an 8-in. clear height of screen. This arrangement increases the life of the screen almost one-half.

The weights of the various parts of a new stamp are:—Shoe, 150; boss-head, 250; stem, 375; tappet, 135; total, 910 lb.

Ten cams with Canda fasteners are on one shaft; the cam-shafts revolve in babbitted boxes without caps, which simplifies the re-babbitting and lubrication. The stamps drop 97 times

per minute through from 7 to 8 in. distance, the order of drop being 1, 3, 5, 2, 4.

At present chrome-steel shoes and dies are mainly used, but other materials also have been employed both in practice and for experiment.

The results of the work of the shoes and dies during a period of 10 months are as follows:

	Length of Life.	Quantity Crushed.	Cost per Ton of Ore.
	Days.	Tons.	Cents.
Chrome-steel shoes, . .	90–95	250	4.46
Cast-iron shoes, . . .	35–40	105	4.95
Chrome-steel dies, . . .	90	240	2.90
Wilson dies, ...	100	270	3.03
Cast-iron dies, . . .	35	100	3.28

Comparative results of work of the shoes and dies are still being obtained.

The battery-solution is kept at a strength of from 1.2 to 1.3 lb. of KCN, and a protective alkalinity corresponding to from 0.8 to 1 lb. of NaOH per ton of solution. Two solutions only are used in the mill, the battery-solution as noted above, assaying about 50 cents in gold per ton, and the barren solution having a strength of from 1.5 to 1.6 lb. KCN and a protective equivalent of from 1.0 to 1.2 lb. of NaOH per ton of solution.

The lime, which is fed with the ore into the battery, is of the average composition of CaO, 92.5; MgO, 0.5; $Al_2O_3 + Fe_2O_3$, 2.0; insoluble, 1.5; and $H_2O + CO_2$, 2.5 per cent.

The stamp-duty per day was not high, averaging 2.66 tons during the latter 6 months of 1903, and 2.80 tons for the first 5 months of 1904. During May, 1904, the stamp-duty was increased to 2.96 tons per day. The quantity of solution going to the battery was between 4 and 5 tons to 1 ton of ore, which, in connection with the hardness and compactness of the ore, accounts largely for the small stamp-duty.

The stamps deliver a product carrying about 60 per cent. of sands and 40 per cent. of slimes. The latter being that portion of the ore the addition of which to water will render it muddy; sands, no matter how fine, leave the water clear; and, because of this sharp line of distinction, the separation of

these two products in the laboratory can easily be made in a gold-pan.

The loss of time in running the battery during the past 10 months, working with a full shift, was: Extraneous causes, 4.20; general mill-repairs, 0.41; engine shut-downs, 0.07; sand-pumps, 0.16; cone system, 0.06; general battery-repairs, 0.35; lack of iron, 0.30; renewing iron, 0.18; total, 5.73 per cent. Of this total loss it is seen that but 0.83 per cent. of the full running-time was due to the stamps proper, which is equivalent to about 6 hours shut-down per month.

Separation.—The separation of the slimes from the sands is one of the most, if not the most vital factor, in the wet-crushing process. Several systems were formerly tried, but they proved to be inadequate. This present method of separation differs from the classification made in concentration in that only a single separation of sands and slimes is made. The system used and elaborated by Mr. C. W. Merrill, at the Homestake mill[1], of using successive cones for throwing off the slimes, and finally using an upward stream for producing a clean sand, seems to be the simplest and most efficient and is now in use, in a more or less modified form, in all of the wet-crushing mills in the Black Hills.

We have found it to be advisable in the separation to make a clean sand rather than a clean slime; a charge of sand with 5 per cent. or more of slimes giving us a low leaching-rate; therefore we are making this clean sand at the expense of throwing some sands into the slimes; these sands are, however, very fine, and cause no trouble in the slime-department and assay, after treatment the same as the slimes proper.

The pulp that the battery delivers flows to two Frenier sand-pumps, 54 in. by 10 in., making 19 rev. per min., and is raised 20.5 ft. to a regulating-box, 3 ft. by 6 ft. by 3 ft. deep, for delivery to two upper cones. This box is intended primarily to take care of the intermittent discharge of the sand-pumps and is provided with a screen to catch any foreign substance, thus avoiding the choking of the cones.

These two upper cones are of simple construction, being 42 in. in diameter at the top and having vertical sides for 12 in.

[1] *Trans.*, xxxiv., 585.

down, at which point the cone starts at a 60° slope, ending in a sorting-column 6 in. in diameter with a 2-in. discharge at the bottom provided with a cock.

The partly cleaned sands, containing from 25 to 30 per cent. of slimes, discharge at the bottom of the two upper cones, then combine and flow to one lower cone, of the same size and pattern as the upper ones, but provided with an upward current of solution; this upward current is taken, after partial settling, from the battery-solution stock-tank under a variable head of from 5 to 20 ft. While an absolute head is preferable, the head employed at present is kept fairly constant at about 15 ft. above the overflow at the cone, and gives practically no trouble. At rare intervals only does this head vary more than a foot either way.

The sands, discharging at the bottom, contain from 1 to 2 per cent. of slimes and pass direct to sand-vats.

The slimes-overflow from the three cones contain from 15 to 25 per cent. of sands, of which only a small portion will stay on a 150-mesh screen.

The products going to sand- and slime-vats from the cones amount each to approximately 50 per cent. of the original ore; the average, since the cone-system was installed 11 months ago, has been 49.03 per cent. to the sand-vats and 50.97 per cent. to the slime-vats.

Treatment of the Sands.—The clean sands from the lower cone, issuing with from 2.5 to 3 parts of solution to 1 part of sands, flow through a launder having a grade of 7 in 100, to the distributor over the sand-vats. The quantity of solution with the sands coming from the cone is not sufficient to carry them through the launder and keep the distributor open and running; and, in order to overcome this hindrance, enough solution is added in the launder to bring it up to at least five parts of solution to one of sands. The distributor is of the Butters' type, having 6 arms, ball- and roller-bearing.

The sand-vats, of which there are 6, are 30 ft. in diameter, 6 ft. deep, and have a lattice-filter frame; 8-oz. duck cloths are used on top of cocoa-matting; the 8-oz. duck has a life of from 8 to 10 months and has been found to be more satisfactory and economical than the heavier grades. The sand-vats hold 140 tons and are filled in about 60 hours.

The method of filling the sand-vats through a vat full of solution has been discarded in favor of "dry-filling," that is—the vat contains no solution when starting to load, and all solution coming in with the sands is allowed to drain off as rapidly as it enters, thus keeping practically a dry surface on top of the sands. This method of filling gives a better leaching-product since the slimes with the sands are evenly distributed through the charge, a result which is not obtained in filling through a vat full of solution. It also gives a more porous charge; the average weight of 1 cu. ft. of sands, as filled into the vats, being 93 lb. (calculated from the last 11 months run). The specific gravity of the original ore averaged 2.7.

As soon as the vat is filled, it is leveled with a stream of solution from a hose under a low head, and battery-solution is run on for a period of 10 days on the average. The small amount of slimes in the battery-solution forms a thin coating on top of the charge and requires an occasional light raking in order to maintain a satisfactory rate of leaching. The battery-solution is followed by barren solution for about 6 days more; the vat is then allowed to drain, and 15 tons of wash-water are passed through; the sands, now ready for sluicing, require from 100 to 150 tons of water for this purpose. The sand-launder has a minimum grade of 8 in 100, which is necessary in order to keep it open.

An average of a large number of sand-vats gives 900 tons of battery-solution and 450 tons of barren solution for one sand-vat treatment (exclusive of the solution filtering through the charge during the filling, which amounts approximately to 700 tons). This large quantity of solution, being nearly 10 tons to 1 ton of sands, together with a total treatment-time of about 16 days, is by no means excessive, for the reason that experiments show as large a volume as possible of weak solution should be kept constantly leaching through the charge.

Treatment of the Slimes.—The overflow from the cones passes to two loading-vats which are filled alternately. There are eight slime-vats (including the two for loading), 24 ft. in diameter and 12 ft. deep, all connected to two No. 4 centrifugal pumps provided with hard-iron linings which can deliver to any one of the slime-vats. The centrifugal pumps have a 4-in. suction and a 4-in. discharge, running at 550 rev. per min., and handle

50 tons of wet pulp per hour. The stuffing-box of the centrifugal pumps is supplied with solution or water under pressure to avoid undue wearing of the shaft and the cutting out of the packing.

Each loading-vat is provided with a partition through the center extending to within 30 in. of the bottom; this arrangement allows the slimes to settle sufficiently so that the clear solution may be decanted from one side of the partition while the vat is being filled on the other side. The time of filling a vat is 12 hours; the stream being then turned into the other loading-vat.

The slimes passing into the loading-vat have 12 tons of solution to each ton of dry slimes; and during a loading of 12 hours about 150 tons (or one-half) of the incoming solution are decanted sufficiently clear to render objects visible through a distance of 2 or 3 feet. This loading-vat, just filled, is decanted as closely as possible, and transferred by the centrifugal pump to vat No. 1, forming one-half of a charge, barren solution being added at the same time; the material in the second loading-vat, after decantation, is transferred to vat No. 2 as one-half of the charge with barren solution; and upon the decantation of these two vats, the contents are combined, constituting the full charge, which is pumped to a third vat; two more transfers and dilutions are then given with barren solution, and, finally, one with water. After each transfer and dilution, several hours of agitation are given by pumping out from the bottom and discharging into the top of the same vat. For the past 11 months the average time of agitation for each charge was 13.87 hours during dilution-treatment and 1.97 during wash-water treatment.

It will be noted that the two largest dilutions are obtained on the half charges when the contained solutions are the richest. A charge of 60 tons of dry slimes from a 24-hour run gets practically the following dilutions with barren solution:—

First half charge, ...	30 tons, 1 dilution of 100 tons.
Second half charge, . . .	30 tons, 1 dilution of 100 tons.
Full charge, ...	60 tons, 3 dilutions of 65 tons.

This treatment makes a total dilution of 395 tons, or a little more than 6.5 tons of barren solution to 1 ton of dry slimes, the

actual figures for the past 11 months being 6.57 tons of barren solution and 0.93 tons of water per ton of dry slimes. Theoretical calculations on this amount of dilution (on the assumption that the total extraction has taken place before the first decantation, and taking the value of the barren solution at 10 cents per ton), the dissolved gold passing out with the slime-tails should assay from 12 to 20 cents per ton of solution, starting with a head-solution of a value from $1 to $2; however, the extraction has not all taken place, but continues slowly throughout the entire treatment, and for a period of six months, the solutions finally going out with the slime-tails showed an average value of 46.1 cents per ton, with an average value of $1.80 for the head-solution, and a value of 10.6 cents for the barren solution. During a period when precipitation was giving us trouble, the value of barren solution averaging 26.7 cents, the value of the head-solution averaged $1.48, and that of the solutions passing out with the slimes averaged 49.8 cents per ton.

The decantations are brought down to a pulp containing from 55 to 60 per cent. of moisture (50 per cent. moisture being equivalent to equal weights of dry pulp and solution). After the decantation of the wash-water, the top layer of the thinner slimes is drawn off, and thrown back to the charge next following. In this way we obtain drier slimes going to waste, averaging, for the past 10 months, 47.0 per cent. of moisture.

While the decantation-process is far from ideal, it is the simplest one at present at our command. The results we have obtained from it have been more satisfactory than experimental tests with other processes; yet it has openings for improvement, and we are now contemplating several changes by which we hope to decant to a drier pulp; and by giving more agitation not only will the final loss of soluble gold be brought to a lower point but the process as well will be simplified.

Battery-Solution.—This solution, of which 1,100 tons per day are used, is pumped by a Prescott 10-in. by 7-in. by 12-in. duplex pump to two stock-tanks, 16 ft. in diameter and 16 ft. deep, situated at the top of the mill. The distribution from the tanks is approximately as follows:—battery, 500; sand-vats, 350; cones, 150; and launder, 100 tons.

Precipitation.—Only the richer solutions from the sand-vats

pass to the gold-tank for precipitation, the balance of the sand-vat solutions, and all the decantations from the slimes, passing to the battery-solution sump.

All standardizations of solutions are made in the gold-tank, thereby obtaining the benefit of the higher strength solution for precipitation,—an arrangement which accounts for the higher strength of the barren solution noted before.

Four 8-compartment iron zinc-boxes of a total capacity of 224 cu. ft., and one 8-compartment iron box of a capacity of 76 cu. ft. are used; hand-cut zinc is used for the reason that it offers a better precipitating-medium than machine-cut zinc; it is, however, true that the consumption of zinc is heavier with hand-cut than with machine-cut zinc. This statement is based upon the actual results of working the two different kinds of zincs, side by side, under precisely the same conditions; and we attribute this result to the hand-cut zinc offering an easier escape for the hydrogen (thus avoiding polarization) than does the machine-cut zinc, because the latter has a smoother surface.

All the zinc used is cut by the mill engineers, who easily cut the 70 lb. per shift that is required, and can cut more than double this quantity if occasion arises.

Data for the last 11 months on zinc-box flow are given in Table II.

TABLE II.—*Data of Zinc-Box Records During* 11 *Months*, 1903–1904.

	Solution.	Zinc.	Quantity of Solution per Day.	Quantity of Solution per Day per Cubic Foot of Zinc.
1903.	Tons.	Cu. Ft. Content.	Tons.	Tons.
July	10,265	194	331	1.71
August	15,130	188	488	2.60
September	14,130	200	485	2.43
October	14,777	212	477	2.25
November	12,793	200	426	2.13
December	12,501	230	403	1.75
Average during 6 mos. in 1903 (Quantitative)		204	435	2.13
1904.				
January	14,306	230	461	2.00
February	13,696	250	489	1.96
March	14,038	265	453	1.71
April	13,859	237	462	1.95
May	14,753	217	476	2.20
Average during 5 mos. in 1904 (Quantitative)		240	465	1.94

Daily measurements of the zinc-box flow, and assays of the head- and tail-solutions, are taken, so that we are enabled to keep very close check on the extractions; the actual bullion-returns exceeded the precipitation-record by 3.4 per cent. for the last six months of 1903, and by 4.7 per cent. for the first five months of 1904.

Of the barren solution, about 25 per cent. is used for the treatment of sands and the remainder for slimes.

The zinc-consumption for the past 11 months is detailed in Table III.

TABLE III.—*Consumption of Zinc During* 11 *Months*, 1903–1904.

	Quantity per Ton of Ore Treated.	Quantity per Ton of Solution of Zinc-Box Flow.
1903.	Pounds.	Pounds.
July	1.86	0.310
August	1.08	0.291
September	1.50	0.316
October	1.15	0.267
November	1.37	0.317
December	1.19	0.300
Average during 6 months in 1903. (Quantitative)	1.33	0.298
1904.		
January	1.48	0.309
February	1.17	0.242
March	1.37	0.329
April	1.19	0.285
May	1.27	0.309
Average during 5 months in 1904. (Quantitative)	1.34	0.295

Cleaning Up.—A clean-up is made twice a month; and, in starting, the flow of one box is shut off and water is passed through for 15 minutes, which is sufficient to replace practically all the cyanide solution. The zinc from the first compartment is washed in the box and removed, the water being transferred to the second compartment, bailing out as close to the precipitated material as possible; the plug in the bottom is then opened and the remaining water, together with some of the precipitates, flows direct to the acid-tank. The precipitates remaining are placed in a tub and carried to the acid-tank. The compartment is washed out with a little water, the plug and screen are replaced and the zinc is replaced. The other com-

partments are cleaned in the same manner, except that the zinc is continually moved to the head-compartments until they become filled. As soon as a head-compartment is filled, the flow of solution is slowly started in order to avoid undue exposure to the air. The water from the last compartment is poured into an adjoining box. One man, in this manner, can clean up the five boxes in 12 hours.

The precipitates are allowed to settle in the acid-tank until practically clear water remains on top; it is then siphoned direct into the waste-sump. When siphoned down as closely as possible (the larger portion of the surface of precipitates being exposed), concentrated sulphuric acid is added and stirred just fast enough to avoid boiling over, the brisk action of the acid raising the temperature. The stirring is done by hand, and no ill-effects have been experienced from the fumes, even with the acid-tank open. The tank, however, is situated in a well-ventilated portion of the precipitating-room. The acid-treatment is accomplished in about an hour, at the expiration of which the tank is partly filled with water and filtering is begun at once; no further washing is now attempted; complete washing, in order to avoid the subsequent production of matte, cannot be done, because of the presence of calcium sulphate; furthermore, as explained later in this paper, the production of this matte has become a desirable feature with us.

A vacuum-filter is used for filtering the precipitates, which are then taken to the melting-room and dried in a muffle-furnace.

The melting is accomplished with a flux, composed of dried precipitates, 10; sodium bicarbonate, 4; borax-glass, 1; sand-tails (60 per cent. available SiO_2), 1.5; fluorspar, 0.2 parts, which gives good satisfaction; the charge melts easily and quietly and gives a clean and liquid acid-slag. The melting is done in a No. 200 crucible and forced draft is used; no skimming is done, since the losses from dusting, in recharging the pot after the skimming, are too heavy.

The bullion goes to the U. S. Assay Office at Deadwood and the slags are shipped to smelting-plants at Denver.

Formerly, it was customary to add wrought-iron scrap to the melt for the purpose of impoverishing the matte, but later developments, giving us an excess of silver above gold, have led

us to flux without iron, thus producing a matte very rich in silver and carrying as high as $10 per pound in gold. This matte is re-melted with sand, a small quantity of flux, and about 10 per cent. of its weight in iron, giving a coppery bullion high in silver and low in gold. The average composition of this bullion is about 700 parts silver and 80 parts gold, which we now treat in a small parting-plant. The bullion, in the shape of a button weighing in the neighborhood of 200 oz., is treated with concentrated nitric acid in a heated porcelain-lined kettle; the operation of parting is not long or tedious, a slime-residue being produced assaying 50 per cent. of gold and 25 per cent. of silver, which is added to the next clean-up. The silver solution is treated with sodium bicarbonate, precipitating the silver as a carbonate, which is roasted at a low-red heat with the formation of silver oxide, and yields a product ready for fluxing and melting into bar-silver.

Just beneath the acid-tank is placed a waste-sump of a capacity of 25 tons of water, into which all of the solutions from the acid-tank and vacuum-filter are allowed to flow. The contents of this sump are found to assay appreciable quantities in soluble gold, but rarely more than a value of $2 per ton. This solution is treated with fine zinc obtained from the zinc-lathe, and sulphuric acid, being well stirred and allowed to settle after the action has ceased. At the expiration of this time it is found that about 90 per cent. of the soluble gold has been precipitated. The sweepings around the zinc-boxes, after a clean-up, are also thrown into this waste-sump. These waste-sump settlings are cleaned up every six months. All ashes of sufficient richness are crushed, and all sweepings from the melting-room are mixed in with slag-shipments.

Cyanide-Consumption.—The consumption of cyanide and lime per ton of ore treated during the past 11 months, both for the chemical and mechanical losses, is given in Table IV.; the mechanical loss being that which passes out in the waste solutions.

TABLE IV.—*Consumption of Cyanide and Lime per Ton of Ore Treated.*

Time.	Consumption of Cyanide.		Consumption of Lime.
	Chemical.	Mechanical.	
1903.	Pounds.	Pounds.	Pounds.
July	0.93	0.67	7.18
August	1.03	0.42	7.91
September	1.07	0.46	7.36
October	1.08	0.38	6.14
November	0.82	0.62	5.48
December	0.89	0.41	5.36
Average during 6 months in 1903 (Quantitative)	0.98	0.49	6.54
1904.			
January	0.52	0.51	5.03
February	0.80	0.39	5.95
March	0.76	0.39	6.34
April	1.04	0.36	5.24
May	1.22[a]	0.59	4.93
Average during 5 months in 1904 (Quantitative)	0.84	0.45	5.49

[a] The increase in the consumption of cyanide during May was due to having raised the solution-strength for precipitation-purposes.

GENERAL.

Although the mill is one of the most modern, the general plan of placing the sand-vats higher than the slime-vats was followed; an arrangement which assumes that clean sands are not to be made and requires an overflow of slimes from the sand-vat to pass to the slime-vats. If clean sands are produced in the separation (this can be accomplished), the logical position for the sand-vats is at a lower level than that of the slime-vats for two reasons; first, the slimes can be delivered at a higher level than the sands; second, the decantation of the richer solutions from the slimes can be run direct on the sand-charges, thus avoiding the unnecessary expense of re-pumping this solution; and, at the same time, the gold-value of the solution going to the battery is kept lower.

Extraction Results.—Table V. gives the data of the extractions from the starting of the mill; the returns for both bullion and slag being quoted separately for the gold.

TABLE V.—*Mill-Results from July, 1903, to May, 1904.*

Time.	Gold.			Silver. Total.
	In Bullion.	In Slag.	Total.	
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Average during the first half of 1903..... (Quantitative)	46.75	1.81	48.56	22.6
1903.				
July.....	50.23	1.39	51.62	26.8
August.....	58.09	0.49	58.58	26.6
September.....	60.26	0.80	61.06	18.6
October.....	63.44	0.49	63.93	17.8
November.....	67.32	2.02	69.34	21.4
December.....	68.23	1.64	69.87	49.2
Average during the second half of 1903.. (Quantitative)	61.15	1.07	62.22	26.1
1904.				
January.....	60.81	1.12	61.93	46.2
February.....	75.16	1.04	76.20	57.3
March.....	76.57	3.15	79.72	42.1
April.....	74.08	1.64	75.72	43.2
May.....	81.81	1.44	83.25	39.3
Average during 5 months in 1904 ..... (Quantitative)	73.81	1.71	75.52	44.3

The extractions shown in Table V., though not high, are regarded as very hopeful, owing to the fairly regular increase. Compared to the difference between head-contents and contents in sand-tails, slime-tails and waste solutions, they show several per cent. higher; cyanide tests on mill-samples have been made regularly for 9 months, and the actual returns exceed the test-extractions by from 5 to 10 per cent. Of the extractions, 47.9 per cent. are obtained in the battery, 27 per cent. from the sand-leaching and 25.1 per cent. from the slimes-treatment during the past 7 months.

Cost of Treatment.—The working-costs vary from month to month, for the reason that all expenses incurred are taken up and charged out at once; and where these expenses cover several months or more, they cause that particular month to have an unduly high cost-sheet; an average working-costs per ton of ore treated at the Maitland mill during the first 5 months in 1904 is given in Table VI. whereby the larger part of this discrepancy is eliminated:

TABLE VI.—*Average Working-Costs Per Ton of Ore Treated at the Maitland Mill, January to May*, 1904.

Department.			
Crusher:	Power	$0.014	
	Labor	0.049	
	Repairs, etc	0.026	
			$0.089
Sampling:	Power	0.004	
	Labor	0.012	
	Repairs, etc	0.001	
			0.017
Stamps:	Power	0.270	
	Sand-pumps	0.023	
	Labor	0.083	
	Repairs, etc	0.073	
			0.449
Leaching:	Pumping solutions	0.050	
	Sluicing sands	0.009	
	Handling slimes	0.022	
	Labor	0.069	
	Repairs, etc	0.018	
			0.168
Chemicals:	Lime	0.027	
	Cyanide	0.305	
			0.332
Precipitation:	Labor	0.011	
	Zinc	0.099	
	Repairs, etc	0.005	
			0.115
Clean-up:	Labor	0.006	
	Acid	0.012	
	Fuel	0.005	
	Fluxes	0.013	
	Crucibles	0.006	
	Repairs, etc	0.002	
			0.044
	Assaying	.....	0.038
	Mill engineers	.....	0.076
	Electric light	.....	0.034
	General expense	.....	0.244
Total expense of milling one ton of ore			$1.606

NOTE.—The treatment cost is increased by the lack of railroad-facilities, all supplies being hauled by wagon from 3 to 5 miles; an average of this hauling cost is 19.6 cents per ton of ore treated.

The expenses of sampling and assaying are equally divided between mine and mill. In the costs of power, the fuel, labor and necessary repairs for the generation of steam are included, the mine and mill being charged equally, which is a very equitable distribution, and the division of the mill-power is based on indicator cards taken from the mill-engine, the cost of a horse-power day averaged 36.3 cents for the past 11 months.

The costs given in Table VI. include every item of expense connected with the operation of the plant, and takes into account also, all renewals and changes made, but does not include depreciation of property or amortization fund.

The clean-up cost per ounce of bullion, based on 11 months production of 23,077 fine oz., was:

	Cents.
Cleaning boxes and acid-treatment,	2.07
Melting-costs,	5.87
Marketing the bullion,	3.46
Total cost per ounce of bullion,	11.40

The labor employed in the mill is classfied as follows:

	Time.	Cost.	
Day-shift:			
Sample-room man	5 hours	$1.25	
Crusher man	12 hours	3.50	
Battery man	12 hours	4.00	
Solution man	12 hours	3.50	
Precipitation man	10 hours	3.00	
Helper	10 hours	2.50	
Engineer	12 hours	4.00	
Total day-shift			$21.75
Night-shift:			
Battery man	12 hours	4.00	
Solution man	12 hours	3.50	
Engineer	12 hours	4.00	
Total night-shift			$11.50
Total labor in mill			$33.25

The men in the mill are advanced from one position to another as occasion arises, which insures a better grade of labor, because they take more interest in their work, and have a more general knowledge of the operations. Moreover, the sudden withdrawal of a man is not so keenly felt. This principle of promotion, which has been in vogue for a year, is productive of excellent results.

Refractoriness of Some American Fire-Brick.

BY R. F. WEBER, ST. LOUIS, MO.

(Lake Superior Meeting, September, 1904.)

THE relation between the chemical composition and the refractoriness of fire-brick has long attracted the attention of manufacturers of fire-brick and others interested in their use, yet but little systematic work has been done along this line, especially in the United States. Analyses of fire-brick have been quoted for years, but fire-tests of brick, except those of durability in practical use, have usually been lacking. Although many tests and analyses of foreign fire-brick have been published, only a few have appeared in the United States; and even in the foreign literature, the references and facts are more or less scattered. The most elaborate foreign paper on the subject is that by Dr. Jochum.[1]

One of the earliest, and perhaps the most detailed piece of work in the United States, is that of F. Platt, which was done for the Pennsylvania Geological Survey, in 1876.[2] His investigation consisted of the analysis and heat-tests of 21 bricks, the testing being made in an experimental shaft-furnace at Harrisburg, Pa. All the bricks were more or less affected at the temperature reached, which was considerably higher than the melting-point of steel. The exact temperature at which individual bricks fused was not ascertained, hence the comparisons quoted in the paper are of value only for the bricks used in the test. The next comprehensive investigation, made by the New Jersey Geological Survey,[3] deals chiefly with fire-clays; analyses and refractory tests having been made of 123 clays, kaolins, feldspars and fire-sands. Here, again, there was no accurate means of measuring the temperature, the only measurement of

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[1] *Thonindustrie-Zeitung*, xxvii., p. 764 (April 30, 1903).

[2] *Second Geological Survey of Pennsylvania, Report MM*, p. 270 (1879).

[3] *Annual Report of the State Geologist of New Jersey for the year* 1880.

the intensity of the heat obtained being from the fusion-point of platinum. Chief among the recent work in this field is that of Prof. Heinrich Ries.[4] The main point noted in his report is that the fusibility of the New Jersey brick was influenced largely by the percentage of silica present, but also, in part, by the texture of the clay; it being found that the fusion-point of almost any of the New Jersey fire-brick could be reduced four or five cones by grinding the brick sufficiently fine to pass through a 100-mesh sieve.

The results presented in this paper have been obtained in testing a series of fire-bricks from different parts of the country, in order to determine the relations of their chemical composition, refractoriness and texture.

Tests that have been made in other countries, as well as in this country, indicate clearly that the variation in refractoriness of fire-bricks depends both on the chemical composition and the texture, although the importance of the latter is not always recognized. The results I present refer to tests made on 44 fire-bricks, representing different localities in the United States. The fusion-point of each of these bricks was determined as closely as possible, also a partial chemical analysis which, though incomplete, is sufficiently valuable for purposes of discussion. The substances determined were total silica, alumina, ferric oxide and titanium oxide. The difference between the aggregate percentages of these components and 100, represents the percentage of alkaline earths and alkalies.

The importance of the estimation of titanic acid has been pointed out by many, including Seger, Vogt[5] and Ries[6].

I have considered first the method of analysis, then the fire-test, then a discussion of the results obtained and, lastly, a comparison of a number of foreign and American analyses and fire-tests.

Methods of Analysis.

The methods mainly used in my experiments were those suggested by Richardson and Hillebrand for the analysis of cement.[7] A "fair" sample of the brick (usually about 60 g.) as

[4] *New Jersey Geological Survey*, vi., p. 71. Abstracted in *Trans.*, xxxiv., 254.
[5] *Thonindustrie-Zeitung*, xxvii., p. 1247 (July 2, 1903). [6] *Op. cit.*
[7] *Journal of the Society of Chemical Industry*, vol. xxi., p. 80 (1902).

prepared for assaying, was ground to pass a 100-mesh sieve, and then sampled down to from 3 to 5 g. A portion of this sample, ground in an agate mortar, was taken for the analysis, and was fused in the usual manner with sodium and potassium carbonates. A double evaporation was made for silica and a double precipitation for iron and alumina. The combined iron and aluminum oxides were fused with $KHSO_4$ in the crucible containing the residue from the hydrofluoric acid correction for silica; the fusion was dissolved with hot 5-per cent. sulphuric acid; the iron was reduced by H_2S and determined, after filtering off the precipitated platinum, by titrating with $KMnO_4$. The titanic acid was determined by Weller's method,[8] and the alumina by difference. The apparatus used for the colorimetric determination was modeled on the lines of that described by Cameron and Failyer, of the Bureau of Soils, U. S. Department of Agriculture.[9] Instead of having a cylinder with side-arm and tubing, however, two Hehner cylinders, with glass-stoppered side-arms, were used. In connection with the determination of titanic acid, it should be mentioned that considerable difficulty was experienced in getting the proper strength of the standard solution. The titanic acid (obtained from a German firm) used to make up the solution was found altogether too impure to allow of a definite quantity being weighed out and the strength of the solution calculated therefrom. Several gravimetric methods were tried, the one finally selected being that of Gooch.[10] Even by this method, it was only after repeated fusions that the titanium oxide was obtained pure, the difficulty being in removing the manganese and iron from the residue.

Fire-Tests.

The fusion-point was determined by testing pieces of the fire-brick in a Deville furnace. Unless a brick is exceedingly fine-grained, the entire mass does not fuse at once; nor does even a large part of it do so. In any brick there is usually found a large percentage of very coarse angular or rounded grains, ranging from 1 to 5 mm. in diameter; and consisting

[8] *Bulletin* No. 176, p. 67, U. S. Geological Survey.

[9] *Journal of the American Chemical Society*, xxv., p. 1066 (1903).

[10] *Some Principles and Methods of Rock Analysis*, by Hillebrand, *Bulletin* No. 176, *U. S. Geological Survey*, p. 71 (1900).

generally of flint-clay, quartz, or "grog" (German chamotte). Surrounding these grains is a more or less fine-grained matrix, which usually is the first to fuse. If the percentage of the matrix in the brick be large the brick may lose its shape rapidly, but if the percentage be small the brick may hold its shape for some time. The fusion-point of the brick was generally taken as the point at which the matrix (and in some cases the larger grains) became sufficiently softened to cause the brick to lose its shape. In many instances the larger grains still retained their identity even after the brick softened down to a more or less viscous mass. In judging the refractoriness of the bricks, therefore, it should be remembered that the fusion-point, as here given, does not necessarily mean that the brick is viscous at that point; in fact, in most cases the brick will stand a higher temperature before it becomes thoroughly viscous.

Analyses and Fusion-Tests.

The results of the examination given in Table I. shows the analysis, the number of the cone of fusion, and the texture of the fire-bricks examined, arranged according to the degree of refractoriness, those bricks having the highest fusion-point being given first. Bricks having the same fusion-point are arranged in descending order according to the percentage of fluxes,—the sum of the iron oxide, titanic acid, alkaline earths and alkalies. Titanic acid is placed among the fluxes, since recent investigation has shown that even 2 or 3 per cent. of this oxide added to a high-grade kaolin will lower its fusion-point two or three cone numbers.[11]

The cone-temperature of fusion was not accurately determined for bricks of a refractoriness less than cone No. 26 or 27. They were thoroughly fused at cone No. 27, but the precise point at which the matrix fused sufficiently for the brick to lose its shape (which, as previously stated, was taken as the fusion-point) was not ascertained. From all appearances, however, the fusion took place between cones No. 24 and 26.

[11] H. Ries, *Trans.*, xxxiv., 254, and *Geological Survey of New Jersey*, vi., p. 71 (1904).

TABLE I.—*Composition, Fusion-Point and Physical Properties of Several U. S. Fire-Bricks.*

No. of Sample.	Locality.	SiO_2	Al_2O_3.	Fe_2O_3.	TiO_2.	Alkaline Earths and Alkalies.	Sum of Fluxes.	Cone of Fusion.	Texture.
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	No.	
1.....	Missouri.	51.59	38.26	1.84	1.97	6.34	10.15	32 to 33	Fairly uniform, angular flint-clay particles, constituting body of brick. Largest pieces 5 to 6 mm. in diameter. White.
2.....	Kentucky.	54.90	38.19	2.18	1.55	3.18	6.91	32 to 33	Coarse-grained; angular pieces of flint-clay as large as 9 mm. Average 4 to 5 mm. Light buff.
3.....	Pennsylvania.	53.05	41.16	2.65	1.80	1.34	5.79	32 to 33	Coarse, angular flint-clay particles, varying from 1 to 5 mm. in diameter. Average 4 to 5 mm. Buff.
4.....	Colorado.	93.57	2.53	0.62	0.27	3.01	3.90	32 to 33	Fine-grained quartz particles. Largest 2 to 3 mm. in diameter. White.
5.....	Kentucky.	44.77	43.08	2.78	2.54	6.83	12.15	31 to 32	Medium grain; flint-clay particles, fairly uniform in size, 3 to 4 mm. Light buff.
6.....	New York.	68.70	20.75	1.20	5.54	3.81	10.55	31 to 32	Coarse grain; quartz particles, 4 to 5 mm. in diameter, forming about 50 per cent. of brick. White.
7.....	Pennsylvania.	49.44	40.94	3.68	2.23	3.71	9.62	31 to 32	Large, angular pieces of flint-clay scattered irregularly throughout brick. Largest 12 mm.; average 6 mm. Buff.
8.....	Colorado.	52.06	40.27	3.47	1.28	2.92	7.67	31 to 32	Medium grain; quartz particles, 1 to 2 mm. in diameter, abundant. Speckled gray-white.
9.....	Missouri.	56.14	39.81	1.84	1.08	1.96	4.88	31 to 32	Angular flint-clay particles, fairly uniform in size. Largest 5 to 6 mm. Light buff.
10	Kentucky.	58.45	34.15	3.53	1.89	6.98	12.40	31	Coarse, large angular flint-clay particles, 4 mm., constitute body of brick. Buff.
11.....	Kentucky.	56.09	31.78	3.16	1.98	7.04	12.13	31	Like No. 10, but average size of particles slightly greater.
12.....	Kentucky.	54.41	36.20	2.10	2.13	5.16	9.39	31	Flint-clay particles, 2 to 10 mm. in diameter (average 5 mm.), scattered throughout. Reddish-brown matrix.
13.....	Kentucky.	50.31	41.21	2.55	2.45	3.48	8.48	31	Medium grain; angular flint-clay particles, 1 to 8 mm. in diameter, forming about 50 per cent. of brick. Buff.
14.....	Kentucky.	55.90	35.71	2.34	2.17	3.88	8.39	31	Large angular particles of clay-flint, varying from 2 to 10 mm. in diameter, forming 60 per cent. of brick. Buff.

TABLE I.—*Continued.*

No. of Sample.	Locality.	SiO_2.	Al_2O_3.	Fe_2O_3.	TiO_2.	Alkaline Earths and Alkalies.	Sum of Fluxes.	Cone of Fusion.	Texture.
		Per Cent.	Per Cent.	Peer Cnt.	Per Cent.	Per Cent.	Per Cent.	No.	
15.....	Ohio.	50.26	41.44	2.02	1.64	4.64	8.30	31	Coarse; contain many flint-clay (and perhaps 3 mm. quartz) particles. Average diameter 4 mm. Cream.
16.....	Ohio.	55.97	36.22	3.44	1.57	2.80	7.81	31	Largely composed of big, angular flint-clay particles. Average diameter 7 mm. Buff.
17.....	Ohio.	57.28	35.23	2.64	1.25	3.60	7.49	31	Flint-clay particles. Average 2 to 4 mm. in diameter. Throughout red-brown matrix.
18.....	Ohio.	50.82	43.51	1.71	1.80	2.16	5.67	31	Coarse, large quartz particles. Average 4 mm. in diameter. Cream.
19.....	Kentucky.	54.71	33.68	2.55	2.02	7.04	11.69	30 to 31	Medium grain; flint-clay particles. Largest 4 mm. Average 1 to 2 mm. Forming 50 — 60 per cent. of brick. Buff.
20.....	Ohio.	59.88	33.35	3.31	1.37	2.09	6.77	30 to 31	Large, angular pieces of flint-clay constitutes greater part of brick; 1 to 10 mm. in diameter. Average 5 mm. Buff.
21.....	Pennsylvania.	50.38	39.81	2.34	1.81	6.66	10.81	30	Medium grain; angular flint-clay particles. Average 3 to 4 mm. Form about 60 per cent. of brick. Buff.
22.....	Georgia.	65.66	28.77	2.18	1.14	2.25	5.57	30	Quartz grains; small to 6 mm. in diameter in an even, close-grained matrix; not abundant. Cream.
23.....	Illinois.	59.67	33.28	1.98	1.51	4.06	7.55	29 to 30	Large, white, angular flint-clay particles. Throughout a brownish matrix. Largest 8 to 9 mm. Not uniform in size, but averaging 4 mm. Brown.
24.....	Texas.	75.78	21.81	1.39	0.77	0.25	2.41	29 to 30	Fine grain, but with a few large particles of grog. Nearly white.
25.....	Illinois.	60.69	28.01	2.51	1.17	7.62	11.30	29	White, flinty particles, 2 to 5 mm. in diameter. Throughout a reddish-brown matrix, and forming 20 per cent. of brick. Brown.
26.....	Colorado.	72.47	19.78	1.67	0.81	5.27	7.75	29	Medium grain; rounded quartz particles; 1 to 2 mm. in diameter. Light cream.

TABLE I.—*Continued.*

No. of Sample.	Locality.	SiO_2.	Al_2O_3.	Fe_2O_3.	TiO_2.	Alkaline Earths and Alkalies.	Sum of Fluxes.	Cone of Fusion.	Texture.
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	No.	
27.....	Pennsylvania.	52.44	40.26	3.57	1.90	1.83	7.30	29	Coarse, large, angular flint-clay particles. Largest 10 mm. Average 4 mm. Forms about 70 per cent. of brick. Buff.
28.....	Colorado.	72.58	20.88	1.69	1.02	3.88	6.59	29	Small, angular flinty particles; 2 to 3 mm. in diameter. Nearly white.
29.....	Ohio.	69.69	25.72	1.90	1.37	1.32	4.59	29	Coarse; a large portion of brick is flint-clay, varying from small to 9 mm. in diameter. Average 4 to 5 mm. Mostly angular. Buff.
30.....	North Carolina.	72.94	15.95	2.81	1.87	6.43	11.11	28	Fine-grained, with occasional quartz particles of considerable size; one 19 by 9 mm., another 9 by 6, but mostly 5 to 6 mm. Cream.
31.....	Ohio.	67.61	25.66	2.42	1.55	2.76	6.73	28	Medium grain; small quartz particles, quite uniform in size (1 to 2 mm), generously distributed. Buff.
32.....	New York.	77.66	9.87	1.29	3.89	7.29	12.47	27 to 28	Fine; only occasional quartz grains of appreciable size. Buff.
33.....	Pennsylvania.	54.81	34.58	3.39	1.51	5.71	10.61	27 to 28	Coarse; largely composed of white to nearly black flint-clay; fairly uniform in size, about 4 mm. Buff.
34.....	Pennsylvania.	60.99	24.34	2.84	1.92	9.91	14.67	27	Medium grain; largely made up of quartz grains; fairly uniform in size, about 2 mm. Buff.
35.....	Indiana.	69.06	21.67	3.23	0.99	5.05	9.27	27	Very fine grain; few coarse particles. Light buff.
36.....	Indiana.	77.01	15.29	1.89	0.93	4.88	7.70	27	Fine grain: some quartz particles 3 mm. in diameter, but these form small percentage of brick. Cream.
37.....	Ohio.	68.99	24.78	2.00	1.12	3.11	6.23	27	Coarse, large, angular flint-clay particles; fairly uniform in size (3 to 6 mm.), forming 40 to 50 per cent. of brick. Buff.
38.....	Illinois.	75.54	14.50	2.72	2.13	5.11	9.96	26 to 27	Medium grain; small quartz grains (1 to 2 mm.) and some flint-clay, forming about 40 per cent. of brick. Buff.

TABLE I.—*Continued.*

No. of Sample.	Locality.	SiO_2.	Al_2O_3.	Fe_2O_3	TiO_2.	Alkaline Earths and Alkalies.	Sum of Fluxes.	Cone of Fusion.	Texture.
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	No.	
39.....	Illinois.	74.58	16.41	3.06	1.31	4.67	9.04	26 to 27	Rather coarse; about 40 per cent. of brick of coarse particles; some of 9 mm. in diameter, but average about 3 to 4 mm. Buff.
40.....	Pennsylvania.	61.28	27.13	2.90	1.37	7.31	11.58	26	Fine grain; small, white flint-clay particles, not over 2 mm. in diameter and not abundant. Buff.
41.....	Pennsylvania.	74.83	16.40	3.26	0.77	4.74	8.77	26	Medium grain; pieces of quartz with pinkish color and angular flint-clay particles. About 3 mm. in diameter. Buff.
42.....	Alabama.	67.19	25.05	2.83	0.71	4.22	7.76	26	Fine grain; even texture. Few coarse particles. Brown.
43.....	Indiana.	60.76	31.66	5.67	1.58	0.33	7.58	26	Fine grain; some particles as large as 1 to 2 mm. in diameter. Buff.
44.....	Kentucky.	60.58	32.49	2.25	1.69	2.99	6.93	26	Angular, dark-colored, flinty-clay particles. Maximum size 5 mm. Throughout a reddish-brown matrix.

Inspection of the data in Table I. shows pre-eminently the variation in the fusion-point, and the generally high percentage of total fluxes. Of the 44 bricks tested, only 4 stood the temperature of cone No. 33, and 7 fused below cone No. 27. Approximately, one-half of the total number fused below cone No. 30. More bricks correspond to cone No. 31 than to any other one cone. Concerning the second point, the percentage of total fluxes seems in many cases very high. It is generally accepted that both fire-clays and fire-bricks should run low in impurities, preferably less than 3 or 4 per cent., and yet here is one of the best bricks having a percentage of flux exceeding 10 per cent. Moreover, in all the bricks having a refractoriness of cone No. 30 or above (with the exception of No. 5, a silica-brick) the minimum flux-content is 4.88 per cent., the maximum 12.4 per cent., and the average about 8.6 per cent. This high percentage of fluxing-ingredients seemed so remarkable that a careful search among published analyses of fire-bricks was made, and it was found that, in a number of cases, the total percentage of

fluxes is high, even in quite refractory brick, as will be shown later in this paper. Moreover, in most analyses, the titanic acid was not estimated, and consequently the alumina-content appears too high and the total fluxes too low, by the amount of titanium oxide present. The fact that titanium oxide is absent from few clays is now generally recognized, and its influence on the fusibility of a clay has been mentioned above. Concerning the percentage of titanium oxide found in the bricks tested, No. 6 alone is noteworthy, for the reason that it is higher (5.54) than that found in any published analyses of American fire-clays or fire-bricks; but Odernheimer[12] cites clays from the Duchy of Nassau with a titanic-acid content of from 2.3 to 4.6 per cent., and Kovar[13] found clays containing from 1.33 to 10.04 per cent. of TiO_2. Of the American clays reported, two from Clinton county, Pa., show the highest percentages of titanic acid, viz., 4 per cent. and 3.36 per cent.[14]

Aside from the two exceptions (mentioned in this paper later) none of the bricks fusing at or above cone No. 30 has a silica-content exceeding 60 per cent., while in the case of those fusing below cone No. 30, the amount of silica present is rarely less than 60 per cent., notwithstanding the fact that many of the more refractory bricks have an extremely high percentage of total fluxes.

In order to make comparisons and to present these facts more graphically than can be done by a mass of figures, the analyses and refractoriness of the bricks have been plotted and curves made on co-ordinate paper, as shown in Fig. 1.

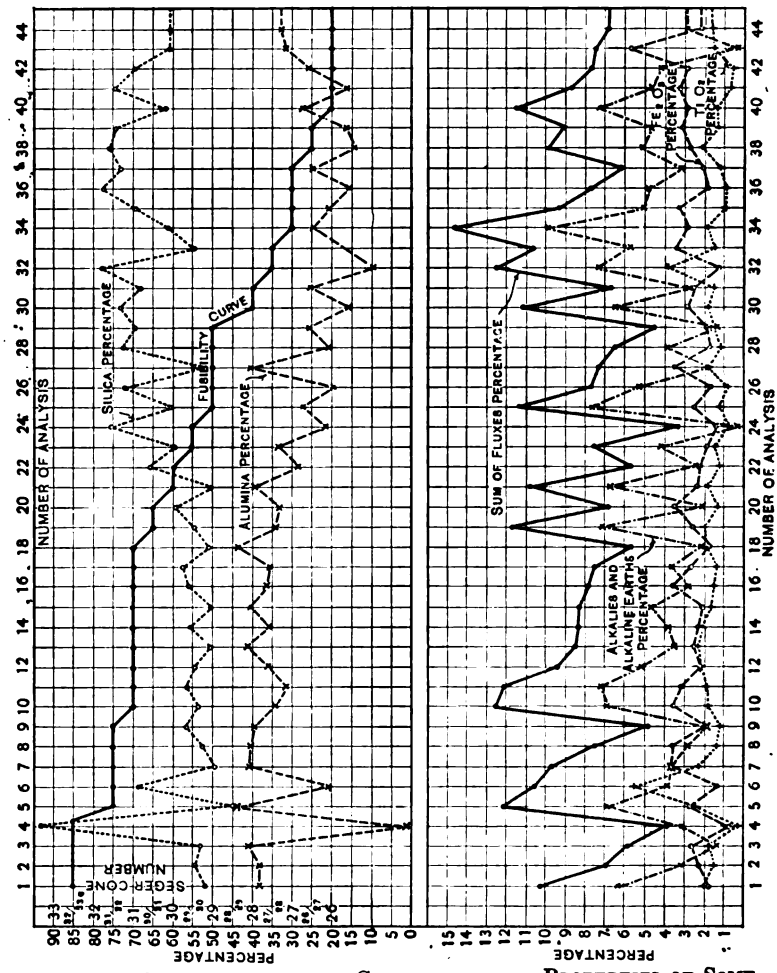
Discussion.

The previously mentioned points are accentuated in Fig. 1, which shows that a line drawn vertically at analysis No. 21 includes, on the left, all bricks fusing above cone No. 30, and on the right, all fusing below it. To the left of this line, the 55 per cent. line would form a fairly-good mean for the silica-content; while to the right, the mean would run close to the line

[12] *Thonindustrie-Zeitung*, xxvii., p. 1475 (Aug. 11, 1903).
[13] *Sprechsaal*, p. 106 (1891); *Thonindustrie-Zeitung*, xxvii., p. 1247 (July 2, 1903).
[14] *Second Geological Survey of Pennsylvania, Report MM.*, p. 265 (1879).

marked 65 per cent. Hence with certain exceptions the silica-content furnishes a rough estimate of the refractoriness of a fire-brick. Moreover, it is apparent at the very start, that the

FIG. 1.



GRAPHIC REPRESENTATION OF COMPOSITION AND PROPERTIES OF SOME AMERICAN FIRE-BRICKS.

percentage of total fluxes cannot form a precise basis for judging the refractoriness of a fire-clay.

Considering Fig. 1 as a whole, if a mean curve for the fusion-points be constructed and a second curve for the silica-content, there is a general rise for the latter accompanying the fall in

the fusion-point, a notable exception being analysis No. 4, which is that of a silica brick. If there were no fluxes present, it would be natural to look for an ascending alumina-curve as the silica-curve descends,—an effect which is generally the case, —the alumina-curve being the image, as reflected in a mirror, of a silica-curve. The exceptions, analysis Nos. 10 and 32, are obviously due to the high percentage of total fluxes. If a mean curve were drawn from the total fluxes, it would be an exceedingly irregular one. The actual curve for the sum of fluxes brings out the effect of a high percentage of total fluxes even in the better grades of brick.

A fair refractoriness of a brick, in spite of a moderately high silica-percentage and a high content of total fluxes, is probably due to the fact that much of the quartz, which is a powerful fluxing-element, is bound up in grains sufficiently coarse to prevent an active fluxing-action. This effect is well illustrated by Samples Nos. 25 and 40. The curves for all the constituents are nearly on a level, yet one has a fusion-point less than cone No. 26, while the other does not fuse until cone No. 29 is reached. Referring to the textures in Table I., No. 40 is of fine grain, containing small, white flint-clay particles, while No. 25, on the other hand, contains quite a percentage of white flinty particles, averaging from 2 to 5 mm. in diameter. Even more strikingly is this point of texture exhibited in Sample No. 6.

Considering the silica- and alumina-curves, No. 35 has almost the same percentage of those constituents as do the others, but, with regard to the fluxes, the difference is in favor of No. 35, yet No. 6 fuses at cone No. 31 to 32 and Sample No. 35 at cone No. 27. The latter sample, however, is of very fine grain, having few coarse particles, whereas probably 50 per cent. of the former sample is composed of quartz particles, from 4 to 5 mm. in diameter. A comparison of Nos. 26, 35, 37 and 42 affords further evidence concerning the influence of coarseness of grain on the fusibility of a fire-brick. In some cases, the high silica-content combined with a high fluxing-percentage (as in Nos. 34, 38 and 39) is sufficiently great to reduce the fusion-point even though the brick contains many grains of medium size.

Another interesting point is that the least refractory bricks

of those tested are not those containing the highest silica-content or the highest percentage of total fluxes—as, for example, Nos. 42 and 43. These, however, are both of fine-grained texture.

A high content of total fluxes does not necessarily indicate a very low refractoriness, for Samples Nos. 5, 10 and 11, with more than 12 per cent. of fluxes, have a fusion-point of, at least, cone No. 31. The refractoriness of these samples was, no doubt, materially assisted by the coarseness of the particles.

It is apparent, from a close examination of Fig. 1, that a high silica-content exerts a more active fluxing-effect than a high percentage of the more fusible fluxes. Following the curve of total fluxes, Sample No. 38 has practically the same percentage as Sample No. 1. There is little change in the various fluxes, but the increase in percentage of silica is very marked, being a rise of 24 per cent. The fall in fusibility can be only partially explained by coarseness of grain, for the texture is much the same, as is shown in Table I. As Sample No. 1 is high in total fluxes, it should be expected that it would be less refractory than one of the same texture containing a lesser quantity of total fluxes. Such a one is Sample No. 37, which, however, becomes viscous six cones below No. 1, an effect which can only be accounted for by the difference in silica- and alumina-content, because the percentage of iron oxide is but slightly greater, and that of titanic acid is nearly 1 per cent. less. Samples Nos. 39, 40 and 41 also seem to indicate that silica exercises the chief rôle in lowering the fusion-point below that of Sample No. 1. The texture cannot here be considered, for both Samples Nos. 39 and 41 are similar in this respect. Nos. 8 and 26 also are of about the same texture and equal percentage of total fluxes, hence the difference in refractoriness must be due to the higher silica-content of one. A comparison of Nos. 3 and 37 points to the same conclusion regarding the greater fluxing-action of a high content of silica.

The value of a chemical analysis for judging the refractoriness of a clay has long been a subject of discussion, and the results presented in this paper seem to emphasize its unreliability for judging the absolute refractoriness of fire-brick. A glance at Samples Nos. 36 and 37 shows how little can be told by chemical composition of even the relative refractoriness of two

bricks. Everything in the analysis seems to indicate a greater refractoriness for Sample No. 37 than for No. 36, yet their fusion-points are the same, even though the texture of No. 37 also indicates a higher fusion-point. In the case of Samples Nos. 27, 32 and 33, the chemical composition is again valueless to account for the reduction of the refractoriness of the bricks; and here, too, the fineness of grain is not responsible. In other cases, as in Samples Nos. 8, 15 and 18, the only explanation offered for the low fusion-point is on the assumption that, as Heintz says,[15] the main mass in highly aluminous bricks is usually much higher in alumina than is the binding-agent used, and from this difference it may result that a brick having a lesser content of alumina may stand better than one having a greater content.

From the foregoing, therefore, it is evident that the refractoriness of a fire-brick depends on the total quantity of fluxes present, the silica-percentage and the coarseness of grain; moreover, chemical analysis alone cannot be used as an index of the refractoriness except within rather wide limits. In the case of clays having no such variation in the size of the grain, it is possible to construct a formula from the chemical analysis which will indicate much more closely the refractoriness of the material.

The results of foreign bricks are shown in Fig. 2, and with one exception (No. X) the samples are of foreign fire-bricks or fire-clays. The locality and the reference to the source of information are given in Table II. Those marked (*) are analyses of German clays used in the manufacture of fire-bricks exhibited at the Berlin Exhibition of 1896;[16] and those marked with (†) are analyses of fire-brick shown at the exhibition at Düsseldorf.[17]

The data given in Table II. refer in nearly all cases to fire-clays, and not to fire-bricks; hence there is no great variation in texture. A great variation in the proportion of total fluxes is, however, apparent here, as in Fig. 1, and it is evident that a high flux-content does not necessarily indicate a clay of low

[15] *Stahl und Eisen*, xvii., pp. 63–64 (Jan. 15, 1897).

[16] *Stahl und Eisen*, xvi., p. 723 (Sept. 15, 1896); Journal of the Iron and Steel Institute, vol. i., p. 357 (1897).

[17] *Stahl und Eisen*, xxii., pp. 1132–1134 (Oct. 15, 1902).

fusibility. Sample H, with a total flux-content of nearly 6, has a refractoriness of cone No. 34, whereas Sample Z, with only half the percentage of fluxes, fuses at cone No. 26 to 27. In the latter case, however, there is a great increase in the silica-content.

TABLE II.—*Names of Foreign Bricks Mentioned in Fig.* 2.

Mark of Sample.	Kind of Material.
A	Clay slate.*
B	Clay slate.*
C	Silica brick.†
D	Clay from Ifö, Sweden.‡
E	Quartzite.†
F	Westerwald clay.*
H	Westerwald clay.*
K	Kahrlich clay.*
L	Rhenish clay.†
M	English Dinas.†
N	Rhenish clay.†
O	Baden clay.*
P	Westerwald clay.*
R	Westerwald clay.*
S	Westerwald clay.*
T	Westerwald clay.*
U	Baden clay.*
V	German Dinas.†
W	Rhenish fire-clay.†
X	St. Louis clay.¶
Y	Bendorf clay.*
Z	Weihersberg clay.*

* *Stahl und Eisen*, xvi., p. 723; *Journal of the Iron and Steel Institute*, vol. i., p. 357 (1897).

† *Stahl und Eisen*, xxii., p. 1132–1134.

‡ *Oesterreichische Zeitschrift fuer Berg- und Huettenwesen*, vol. l., pp. 492–1494.

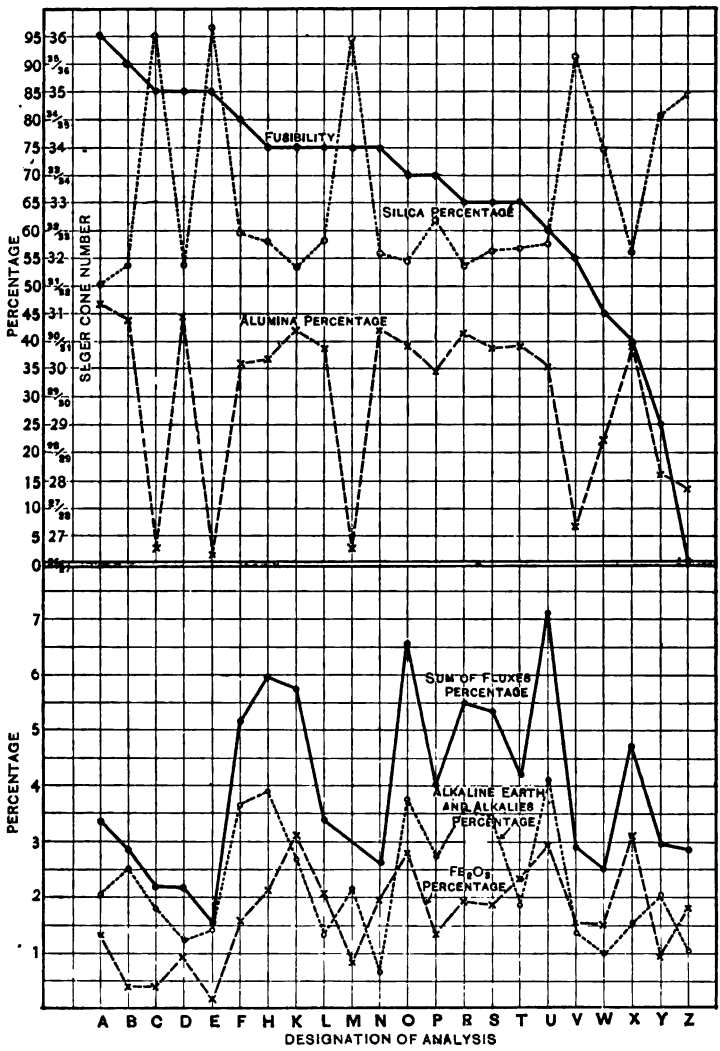
¶ *Thonindustrie-Zeitung*, No. 45, p. 656 (April 16, 1903).

Figs. 1 and 2 show that, above a certain percentage (about 90), the silica does not exercise so powerful a fluxing-action as it does when it is somewhat lower. This result is entirely in accordance with the known facts concerning mixtures of kaolin and silica. Up to about 90-per cent. content of silica, the fusion-point decreases as the percentage of silica increases. From that point, however, the fusion-point rises rapidly with increasing percentage of silica.

A further evidence of the powerful fluxing-action of silica may be obtained by studying the curve of total fluxes, which shows that Samples L and Y are nearly on a level with A;

sample L, however, contains 8 per cent. and Y 30 per cent. more silica, and, as would be expected from what has been shown in

FIG. 2.



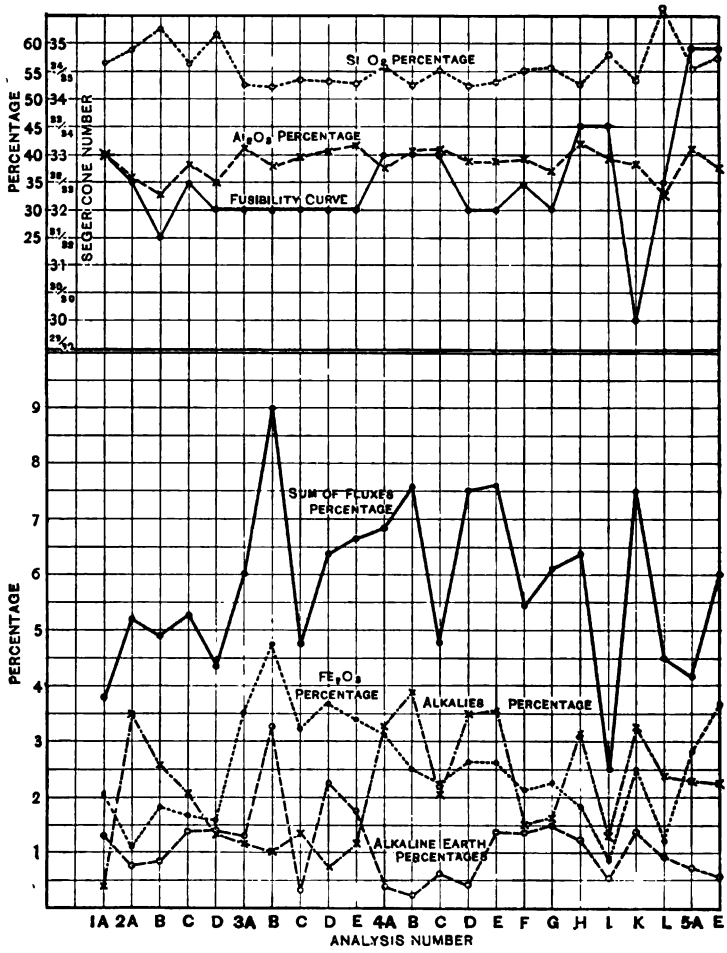
GRAPHIC REPRESENTATION OF COMPOSITION AND PROPERTIES OF FOREIGN FIRE-BRICK.

the case of American bricks, the fusion-point declines from cone No. 36 to cones Nos. 34 and 29 respectively.

With Fig. 2, as with Fig. 1, the fusibility-curve is a steadily

descending one, and, setting aside the silica-bricks, the 55-per cent. line, as before, forms a fairly good mean for those clays exceeding cone No. 30 in fusibility. No mean curve, however,

FIG. 3.



GRAPHIC REPRESENTATION OF COMPOSITION AND PHYSICAL PROPERTIES OF FOREIGN FIRE-BRICK (JOCHUM).

could well be drawn for the sum of fluxes, either with foreign or American fire-bricks.

Fig. 3, taken from Jochum's article in the *Thonindustrie-Zeitung*, previously mentioned, is comparable only with about the first 20 of the American bricks, both in refractoriness and

silica-content. It shows, however, the same variation in the sum of fluxes, and furnishes further evidence that a high flux-content does not necessarily indicate a low refractoriness.

The foreign bricks and clays do not generally show such a high percentage of total fluxes as do the American products, due in part, as before mentioned, to the non-estimation of titanium oxide.

I take this opportunity of acknowledging my indebtedness to Prof. Heinrich Ries for suggestions made during the course of the work.

The Equipment of a Laboratory for a Smelting-Plant.

BY HERBERT HAAS, INGOT, SHASTA COUNTY, CAL.

(Lake Superior Meeting, September, 1904.)

THE following notes describe a laboratory for metallurgical chemistry and technical analysis which I built late in 1903, while engaged, as constructing engineer, in erecting a pyrite smelter at the Afterthought mine, Ingot, Shasta county, California.

This design partly supplements the paper of Mr. C. H. White, which was presented at the Atlantic City Meeting of the *Institute* in February, 1904.[1]

The work of a chemist at a busy smelting-plant differs radically from the work of a student, who takes all possible care to have his analysis exact within hundredths of a per cent., and has plenty of time to devote to the operation. The works-chemist, having the shortest time in which to make his determinations, should have a laboratory arranged as conveniently as possible to save time in carrying out his work. By a conveniently-arranged laboratory, I do not mean an elaborately furnished one; but rooms suitably fitted up and supplied with the necessary apparatus to do the work expected from the chemist within the requirements of technical analyses.

It is not uncommon at large smelters to have the night assayer prepare 30 or more samples for titration, in time for the

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[1] (See page 117.)

chemist to begin work on them at 7 o'clock on the following morning. During the time that the solutions are being heated on the hot-plate, the chemist is weighing out portions of slag-samples which have been taken from each furnace. As a rule, he determines the various percentages of silica, iron and lime in the slag (the values in lead and silver, and, sometimes, that of gold being determined by the assayer), but very frequently the slag is analyzed for other constituents as well; and every 14 days a complete analysis of it is made. The chemist cannot give his entire time to a single analysis until it is completed, for the reason that a dozen or more additional samples come in regularly which have to be analyzed for "insolubles," silica, iron, lime, zinc, sulphur, lead, copper, alumina, manganese, arsenic and antimony.

Every 14 days, both flue-dust and speiss are tested for their copper- and lead-contents, and occasionally a complete analysis of these products is required. The lead-bullion is analyzed for its content of iron, copper, arsenic, antimony and bismuth. In addition to the routine work mentioned above, an analysis of coal or coke for the percentage of ash, sulphur, volatile matter and fixed carbon, as well as a determination of its calorific value, is called for, and occasionally an analysis is made of refractory brick and of gas. The chemist, in order to do all this work must devote his entire time for 10 hours every day. The above-named conditions are not hypothetical, but have actually occurred in my own experience at a large lead-smelter; and I have taken them into consideration in my present design of laboratory.

The chief requisites for a works-laboratory are abundant light and good ventilation, because, even with the best arrangement of hoods and flues, the air is liable to become contaminated by acid fumes.

The laboratory-building is constructed of wood, and is ceiled inside with tongue-and-groove lumber, leaving a 4-in. air-space between the outer and inner walls. The roof terminates in a large ventilator, having windows swinging on pivots over the laboratory portion, while over the assay-office the ventilator has fixed shutters of 1- by 6-in. wooden slats, set at an angle of 45°, with 3-in. spaces between adjacent slats. A chemist's hearth, having a hood, supplies the heat necessary to boil solutions, as

well as for other uses. The main working-table is placed opposite the chemist's hearth, which arrangement allows the chemist to carry on his analytical work and, at the same time, control the heating of the samples in sight on the hearth. The table is equipped on both sides with drawers, closets and shelves; the side nearest the hearth is used for quantitative work only, while on the other is a complete set of the reagents needed for qualitative mineral analysis. A small kerosene-stove, and a copper kettle, placed on the highest shelf of the table give a constant supply of hot distilled water. On opposite sides of the bottom of the kettle are two small bibbs, to which are fastened small rubber tubes, having at each outer end a pointed glass tube, and sufficiently long to reach both ends of the table. A Mohr pinch-cock regulates the flow of water to a nicety, and the 4-ft. head gives sufficient pressure to wash precipitates, filters, etc., thoroughly and quickly. This arrangement avoids the discomfort arising from continued inflation of the cheeks which the ordinary wash-bottle necessitates. A sink of glazed clay with drain-boards is placed at the end of the table.

A table opposite the main working-table is reserved for electrolytic analysis, and is equipped with an accumulator, which is used as the source of current. The accumulator is better than the common cells for the reason that it maintains a fairly constant potential. The current strength and potential are respectively measured by an ammeter having a range of 5 amperes with subdivisions of 0.1 ampere, and a voltmeter with a range of 10 volts, and subdivisions of 0.1 volt. The rheostat consists of a band of German silver placed in the circuit and having a sliding-contact which enables the operator to change the potential. As the electrolytic determinations are limited to copper-mattes, and occasionally leaf-copper for standardizing-purposes, the amperage and voltage best suited to the chemical composition of the matte will be used; and when once ascertained, they will remain practically constant. Luckow's cylinder and spiral are used as electrodes.

A table in front of a window is used for titrations and is provided with burettes, which can be filled automatically. This special type of burette consists of a 2-liter glass bottle with a pressure- and a delivery-tube passing through the cork, the delivery-tube connecting direct to the upper end of the burette.

The pressure necessary to force the solution from the bottle to the burette is supplied by two rubber bulbs having air-inlet valves so arranged that a continuous pressure may be produced. The inner (and larger) bulb is covered with a strong cord-net in order to remove the danger of bursting from too great a pressure. On the table are two closets, having shelves large enough to hold a 2-qt. bottle, and serving the purpose of keeping the solutions in the dark; the closet doors bear the symbols of the respective stock-solutions. In my laboratory I have the following-named stock-solutions,—potassium cyanide, potassium dichromate, potassium permanganate, potassium ferrocyanide, $\frac{N}{10}$ potassium hydroxide, $\frac{N}{10}$ sulphuric acid, copper sulphate, ammonium molybdate. An additional set of these solutions kept in reserve in the dark closet allows a convenient interval of time in which to prepare fresh solutions.

A cupboard is provided in which to store a complete set of the chemicals needed in this line of work, as well as such apparatus and glassware as are only occasionally used.

The balance-room is tightly ceiled, special care having been taken to make it dust-proof. It has three windows on each long side, and a swinging sash-door entrance at the end, which guards against collisions between assayer and chemist when passing through the door-way in opposite directions.

The bench on which the balances rest is supported by wooden posts set in concrete, and extending through the floor into the ground with a 0.25-in. clearance. The scales set on plate-glass, and when once adjusted, they remain level for a considerable time, any shrinkage of the table having no effect on the position of the plane of the plate-glass.

The general arrangement of the laboratory and assay-office is shown in Fig. 1.

The chemist's hearth, a description of which may possess special interest to chemists and metallurgists at places where gas is not obtainable, is illustrated by Fig. 2. This hearth rests on the ground and the flue from it is filled with ashes and earth to within 18 in. of the hot-plate. In the front of the ash-pit, beneath the assay-office floor, is a 12- by 12-in. opening, which is closed with a piece of sheet-iron luted on with clay. The accumulated ashes in the ash-pit are removed through this

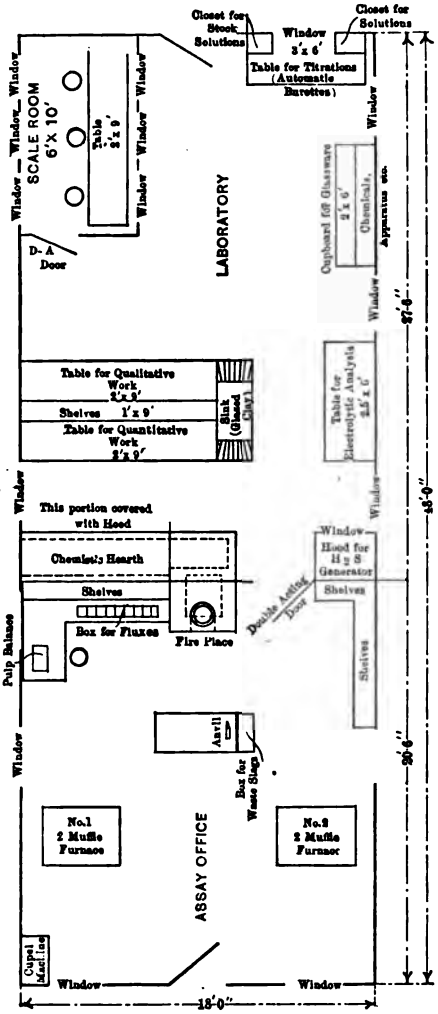
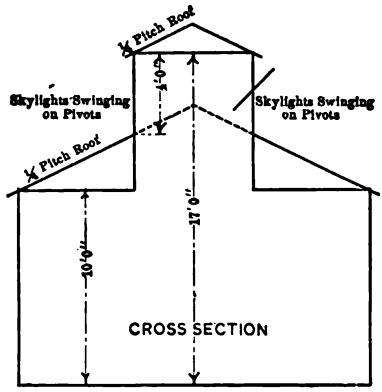


FIG. 1.

PLAN OF ARRANGEMENT

GENERAL ARRANGEMENT OF LABORATORY AND ASSAY OFFICE FOR A SMELTING-PLANT.



opening, thus avoiding their removal through the assay-office. The ash-pit door shown in the drawing is used solely to regulate the draft. Old rails preferably are used as grate-bars.

The front elevation of the hearth is shown in Fig. 2, which includes also the elevation of the fire-place front with the ash-pit door and the feed door. The lines, A B and C D, explain the respective elevations. The walls of the hearth consist of one course of brick, excepting at the stack, which is of a course and a half, and the fire-box, which is of two courses. These walls support the hot-plate having its upper surface 43 in. above the level of the floor. A detailed dimensioned drawing of the hot-plate is given in Fig. 4. The plate is cast in two pieces, having a lap, so that a tight joint may be obtained, and at given intervals ribs are cast as a safeguard against warping. A circular hole, over which the still is placed, is left in the plate. A portion of the hot-plate 2.5 ft. by 6 ft. 8 in. is covered with a hood, which rests on one layer of bricks, except at the hottest parts, where there are two layers in order to protect the wood. The back side of the hood does not rest on bricks, but is separated from the plate by a 2-in. air-space extending the entire length of 6 ft. 8 in. Access to the hot-plate is obtained through two windows, each having 12 lights of glass and hung on butts. The hood is tightly ceiled with tongue-and-groove lumber, and has an 18- by 18-in. wooden chimney, 10 ft. high, to carry off the fumes. The temperature of the inside of the hood and hood-chimney is sufficient to draw in fresh air constantly and thus improve the ventilation of the laboratory.

The great advantage of the hot-plate is its gradual decrease of temperature towards the chimney. The heating of solutions is generally started at the coolest place, and gradually continued toward the hottest part. The heat is diffused over a large area, and is not concentrated at one small spot as is the case with a Bunsen burner, and the boiling over of solutions is thus easily avoided at the expenditure of the least attention and care; thus allowing the chemist time in which to attend to other work. A small uncovered portion, 2.5 ft. by 3 ft. 4 in., is reserved for operations which are preferably conducted in the open air. The fire-place extends into the assay-office, and is closed off

FIG. 2.

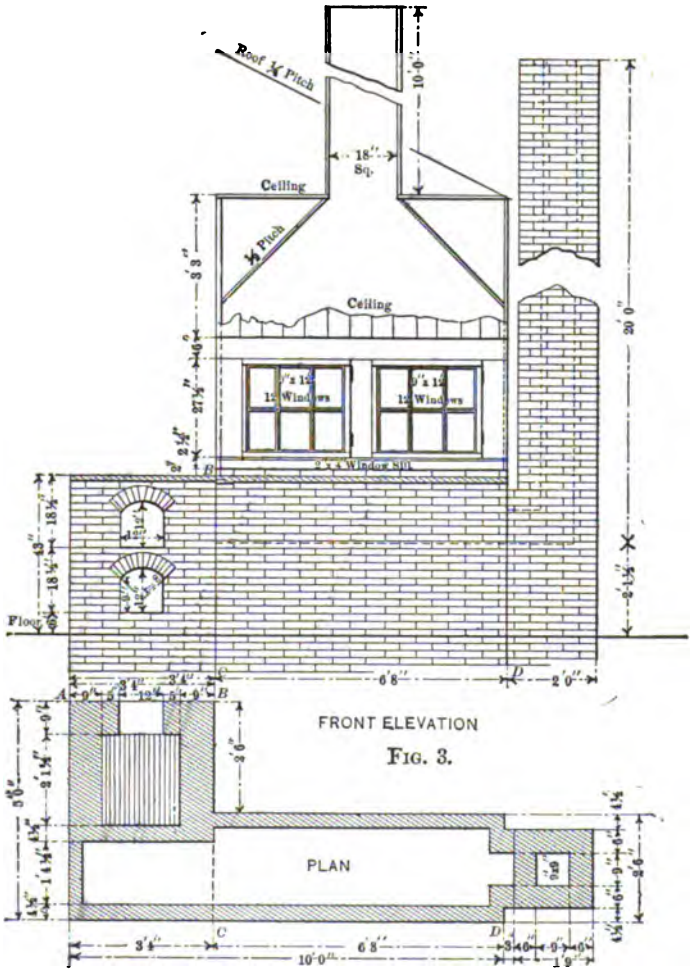
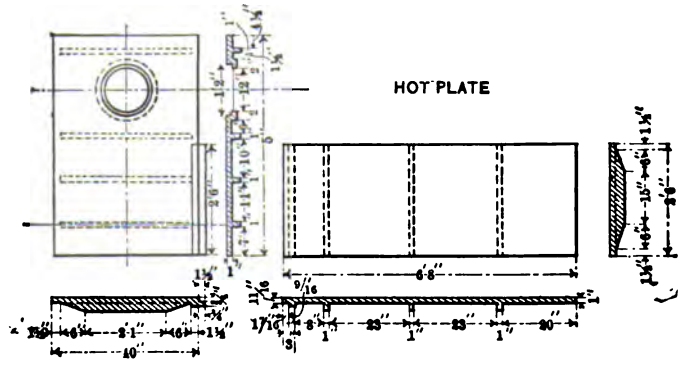


FIG. 4.

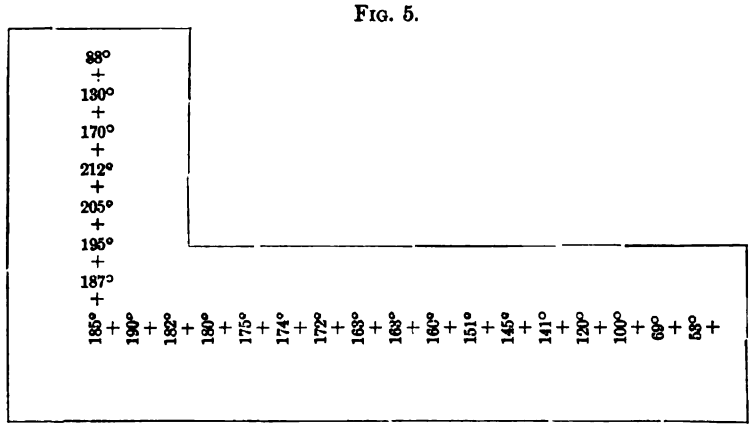


DETAILS OF THE CHEMIST'S HEARTH.

from the laboratory by a ceiled partition, formed by the continuation of the back side of the hood.

A still, placed over the circular hole described above, provides the laboratory with about 14 gal. of distilled water daily. The still now used is called the Cuprigraph Sanitary Still, No. 11,[1] and when once regulated requires very little additional attention. The hearth thus serves two purposes, one, to provide the laboratory with distilled water; and the other, to give the chemist, at the same time, a very efficient way of heating solutions. The distillation of the water utilizes much of the heat, yet the hearth is necessarily wasteful, the flue being too short for economizing fuel. The weekly fuel-consumption, however, is only from one-half to three-quarters of a cord of wood, which, at $4 a cord, is equivalent to an operating-expense of from 28 to 42 cents a day.

FIG. 5.



TEMPERATURES IN DEGREES CENTIGRADE AT DIFFERENT PARTS OF THE HOT PLATE.

Fig. 5. shows the decrease in temperature with increasing distance from the fire-place. The temperatures were taken with a thermometer registering up to 250° C., and, while the readings are not exact, because the hearth was fired intermittently, they convey a fairly true idea of the temperature at different portions of the plate. Generally the hot plate is not used at temperatures as high as those recorded.

[1] Made by the Cuprigraph Company, Chicago, Ill.

The laboratory is also provided with a hood, containing a hydrogen sulphide generator.

The assay-office has two muffle-furnaces, one kept as a reserve, or used when an accumulation of work has to be done. A general fluxing-mixture is kept, which, with slight alterations, can be used to flux ordinary ores.

All the samples are received from the sampling-mill in a ground condition, passed through a 100- or 125-mesh sieve. No grinding or sampling is done in the assay-office. The cupels are made by a cupel-machine.

The cost of the assay-office and laboratory-building (not including chemical apparatus and supplies) is given in Table I., and a detailed analysis of cost of chemist's hearth in Table II.

TABLE I.—*Cost of Assay-Office and Laboratory-Building.*

Common lumber, 6,000 ft. (board measure), @ $16 per M	$96.00
Flooring, ceiling, 2,427 ft., @ $35 per M	84.94
4.5 M. shakes, @ $11 per M	49.50
4 doors	12.50
Locks and butts	8.50
Transom-plates and sash-bolts	4.00
Windows	69.00
Nails	15.00
Labor	185.00
Total	$524.44

TABLE II.—*Cost of Chemist's Hearth.*

Hot-plate, 1,451.48 lb., @ 4c. per pound	$58.06
Freight	7.26
Bricks, 3,000, @ $10 per M	30.00
Mortar and clay	7.00
Bricklayer, 3 days, @ $5	15.00
Helper, 3 days, @ $2.25	6.75
Lumber and other materials	11.50
Carpenter work, 2 days, @ $3.15	6.30
Total	$141.87

My thanks are due to Mr. F. K. Baxter, Jr., who has kindly assisted me in determining the temperature-conditions of the hot-plate.

Biographical Notice of Sir Clement Le Neve Foster.*

BY T. A. RICKARD, NEW YORK, N. Y.

(Lake Superior Meeting, September, 1904.)

CLEMENT LE NEVE FOSTER was born at Camberwell on March 23, 1841, his father being Peter Le Neve Foster, who was secretary of the Society of Arts for 26 years. As a boy of 12 he was sent to school at Boulogne, receiving a preliminary education there; and in 1857, when only 16 years old, he took his degree of Bachelor of Science from the University of France. In October of the same year he returned to London and entered the Royal School of Mines, which was then in Jermyn Street. He distinguished himself by winning a scholarship and a medal, and by securing his degree as Associate of the Royal School of Mines in the abbreviated period of two years. This was followed by a year at the Bergakademie at Freiberg; subsequently he visited several mining districts in Germany and Hungary. On returning to England, in November, 1860, he received from Sir Roderick Murchison an appointment upon the Geological Survey. During five years he was engaged in field-work, first on the Wealden beds in Kent and Sussex, and then in Derbyshire and Yorkshire. At this time his evenings were largely devoted to reading for a degree from London University. In 1861 he matriculated, obtaining the Bachelor of Science in 1863, with honors in biology and a University scholarship in geology; in 1864 he passed the two examinations entitling him to the Bachelor of Arts degree, and in 1865 he received the degree of Doctor of Science.

In 1865 he succeeded Mr. Richard Pearce as lecturer to the Miners' Association of Cornwall and Devon. At the same time he became the curator to the museum of the Royal Geological Society of Cornwall at Penzance, a member of the council in 1873, and a life-member in 1874; he made frequent donations to the museum, besides contributing several valuable papers to

* Published also in the *Engineering and Mining Journal*, June 30, 1904.

SIR CLEMENT LE NEVE FOSTER.

the transactions of that society. This pleasant association was recognized when, in November, 1900, he received the William Bolitho medal in token of his professional eminence and the services rendered by him to the society. In the following year he became president and, in that capacity, had the pleasure of presenting the Bolitho medal for 1902 to his friend and colleague, Richard Pearce.

In 1868 he resigned his appointments in Cornwall to join an exploring expedition sent to examine the mineral resources of the Sinai peninsula, and later in that year he went to Venezuela and made a report on the Caratal gold-field. Returning to Europe, in 1869, he was appointed engineer to the Pestarena Gold Mining Company, and for three years he was resident manager of these mines at Val Anzasca, in northern Italy.

Meanwhile the Metalliferous Mines Regulation Act of 1872 had been passed, and in 1873 he was appointed an inspector to carry out its provisions. As inspector of mines for the district of Cornwall and Devon, he returned to the mining community where he already had so many friends, and he expended much persistent effort to improve old methods of working. This was not done without opposition and some friction on the part of the Cornish miners. Sufficient comment on the usefulness of his official labors is offered by the fact that the average death-rate in his district was reduced from 2 per 1,000 during the first three years of his inspectorship to 1.3 per 1,000 during the last five years. He won the support of the best men in the West Country, and as joint secretary of the Royal Institution of Cornwall, as one of the founders of the Mining Institute of Cornwall and of the Miners' Club and Relief Society, his efforts to benefit the mining industry of the two counties became heartily appreciated. In 1880 he was transferred to the North Wales district, much to the regret of all who had worked with him for the benefit of the West Country. He retained the position of inspector of mines for North Wales until his retirement in 1901.

In 1890 he succeeded Sir Warington Smyth as Professor of Mining at the Royal School of Mines, and this post he held until his death. His professorship was marked by many improvements in the course of instruction; chiefly in a practical direction, as was recognized by a marked increase in the

number of students. He was always busily occupied; in 1894 he became editor of the mineral statistics issued by the Home Office, and of the annual reports on the mines and quarries of the United Kingdom. Even after his resignation as inspector of mines, he was retained by the Home Office to edit these publications, which had become recognized as of the greatest value to all interested in the mining industry. He did a great deal of useful work as a member of Royal Commissions and on Department Committees; he served as juror at the Paris Exhibitions of 1867, 1878, 1889 and 1900, these services becoming fittingly recognized when he was created a Knight of the Legion of Honor. He was also a juror at the Inventions Exhibition and other succeeding exhibitions held in London.

In 1893 he was elected a Fellow of the Royal Society. In 1901 he resigned the position of inspector of mines and in 1903 he received the honor of knighthood at the hands of the King.

During all these busy years he found opportunity for authorship; he was a man of indefatigable energy, and during the intervals of his official labors he contributed papers to the societies of which he was a member. In 1876 he joined with his former colleague, W. Galloway, in translating into English the treatise on "Mining" written by Professor Callon, of the École des Mines, at Paris. This translation continued for a long time to be a standard text-book. He also translated a book in Dutch on the Banka tin-deposits. He wrote the articles on "Mining" in the *Encyclopædia Britannica*. In 1894 his most important book, the *Text-book of Ore and Stone Mining*, appeared; this has been accepted as a text-book in most English-speaking mining schools. Last year the smaller manual, *Elements of Mining and Quarrying*, was published, and met with a cordial reception. He died in London on April 19, 1904.

It is probable that his life was shortened by the episode in the Snaefell mine. This is a lead-mine in the Isle of Man. In 1898 there was a fatal explosion underground, and as inspector of mines he was early on the spot. After the accident he went down with a rescue party, which was overpowered by the fumes and prevented from getting immediate assistance by the cage becoming stuck in the shaft. For a long time they were

in great danger of suffocation, and the clear-eyed courage with which Le Neve Foster faced imminent death was shown by the fact that he made notes of his sensations and of his physical condition during the period spent in awaiting rescue. In a manner this incident typifies his life; there was heroism in the discharge of duty, there was the scientific spirit which wanted to leave a record which might be of material service, and there was a well-ordered intelligence in the way it was done. From the shock to his system he never quite recovered, and it undoubtedly shortened his days, so that in the end he sacrificed his life in the discharge of his duty.

He left a host of friends. A representative Cornishman, Mr. J. H. Collins, says: "By the death of Sir Clement Le Neve Foster the mining industry in general, and West of England mining in particular, has experienced a loss which it will be hard indeed to repair; his versatility was a continual surprise, and his charm of manner endeared him to all who had the privilege of his intimate acquaintance; his memory will long be green in the West country."

His colleague, Professor William Gowland, of the Royal School of Mines, writes: "As a miner his death will be keenly felt by all engaged in British mining, for which he had done so much; while, as a man of kindly heart and good fellowship, his removal from our midst will be deeply deplored by that wide circle of friends who had the privilege of knowing him intimately."

Le Neve Foster was a juror at the Chicago Exposition of 1893, and at that time many on this side of the Atlantic had the pleasure of meeting him. It was then that he became a member of the American Institute of Mining Engineers. That representative American mining engineer, Mr. Hennen Jennings, speaks thus of him: "His brilliant attainments, strong character, modesty, and charm of manner have most deeply impressed me. As a guide and an authority in our profession I could look up to him, yet as a colleague I felt always free to unreservedly express opinions which might differ from his own without fear of giving offense, and I could feel that in him I was gaining a rare friend."

Mr. A. G. Charleton, a colleague of his in the work done by the Institution of Mining and Metallurgy, London, of which

he was one of the editors, says with much feeling: "I am certain that all who knew Sir Clement Foster personally will always treasure the recollection of his manly and kindly personality and friendship. His life, like Alexander Lyman Holley's, should, I think, be an incentive and object-lesson, to lead all of us mining engineers to humbly endeavor to follow the path he so uprightly trod, and to take pride in the advancement of the profession he adorned by his example and teaching."

Yes, that is the story of his life; and the telling of it will have had something far different from a mortuary effect if it serves as a stimulant to high endeavor in the profession to which all of us belong. A career so singularly consistent in its preparation, fulfilment and honorable close, affords strong encouragement to younger men. Men of ability are apt to get what they aim at if they strive consistently; Le Neve Foster always had noble aims and a persistent purpose—to be a useful official, a helpful friend and a successful educator. He was all of these in full measure, and long after his endeavors have ceased, his noble personality will be perpetuated, by better than marble, in the example of a useful life.

The Concentration of Gold and Silver in Iron-Bottoms.*

BY MYRICK N. BOLLES, B.S., NEW YORK CITY.

(Lake Superior Meeting, September, 1904.)

INTRODUCTION.

THE concentration of gold and silver in mattes low in copper, and the subsequent separation and recovery of either or both of these metals, is a question the satisfactory solution of which has long vexed metallurgists. It covers a broad and important department of metallurgical science and brings in its train a series of attendant problems continuing from the initial operations of concentration to the final recovery of "values."

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* This paper was submitted in partial fulfilment of the requirements of the degree of Doctor of Philosophy in the Faculty of Applied Science, Columbia University, New York City; and at the same time was accepted by the American Institute of Mining Engineers for publication in the *Transactions* of that Society.

The particular phase of metallurgy, which perhaps is most vitally associated with this series of problems, is the so-called pyritic smelting for the reduction of gold- and silver-ores. This association follows from the conditions which govern the operation of the majority of such plants and under which they must be expected to remain while they continue to occupy their present and, what seems to be, their natural position in the scheme of metallurgical economy.

The principal object of these plants being, as a rule, the use of low-grade matte as a carrier for either gold or silver, and these two metals being the source of profit, it follows that a matte is usually made which is too rich in these metals to be allowed to follow the ordinary course of treatment to which copper-matte, for instance, is subjected. It may be, and often does happen, that such matte is produced in operations which must be classed under modes of treatment other than pyritic, but these cases are the exception and not the rule. In such cases, however, the subsequent treatment of the matte produced becomes of great importance.

The most advantageous conditions for such pyritic smelting seem to be where refractory gold-bearing ores of a smelting grade occur remote from lead- or copper-deposits, or where the cost and scarcity of such ores prohibit their free use as carriers and collectors of the precious metals.

In such cases to escape the expensive resort of shipping the gold-bearing ores to a more favorably placed plant, recourse is often had to one of the varieties of pyritic smelting if the ore is rich in sulphur; or, if it is oxidized or "dry," sufficient low-grade or even barren pyrites are charged, and the operation conducted quite as in any matting operation in the blast-furnace. In either case, some copper-ore is usually charged and the result is ferruginous matte carrying more or less of the precious metals, as the case may be.

The copper-ore charged in order to produce this matte is in such cases often treated at a loss on account of the expense of shipping it to the plant,[1] hence it is usually the aim of the smelter to produce a product as low in copper as possible.

As yet no satisfactory method of direct treatment of such

[1] F. R. Carpenter, *Trans.*, xxx., 772.

matte (containing less than 10 per cent. of copper, and from 20 to 40 oz. of gold with perhaps 100 oz. of silver per ton) seemed to have been evolved, and for this reason it is found most convenient to dilute it largely by introducing it in small quantities into some one of the more ordinary metallurgical operations which is not over-burdened with an excess of gold or silver.

The usual practice is to ship it to a lead- or copper-smelter where it is charged, either raw or roasted, with a large percentage of the regular charge, and the precious metals eventually recovered, from the resultant metallic lead by the Parkes process, or from metallic copper as vitriolization- or electrolytic-slimes.

The smelting-charge for the treatment outlined above is usually high, in addition to heavy freight-rates for transportation. The copper-content causes the production of lead containing a large quantity of very objectionable copper as an impurity; and a large quantity of copper-matte carrying sufficient lead, so that the subsequent and heavy loss of this latter metal upon retreatment is a serious item to the lead-smelter. On the other hand, it is rather low in copper, and too high in gold and silver, to be acceptable to the copper-smelter. Thus it is a difficult and expensive product to place on the market.

Local treatment as a rule is out of the question, as it is too low in copper and too high in the precious metals either to bessemerize or to concentrate by roastings and smeltings in the reverberatory furnace.

Methods of Direct Treatment.

Direct treatment is the "open sesame" of the situation, and as the problems of initial concentration to the ferrous matte have been fairly well mastered, the recovery of the contained precious metals remains as the residual matter of importance.

To effect the separation of gold and silver from ferrous or cuprous sulphide (leaving out the field of hydro-metallurgy entirely), recourse may be had to two distinct and radically different operations.

1. By the difference in affinity (that is,—to express more definitely, the heats of combination) of the constituent elements for an outside agent (more often oxygen assisted by silica, etc.),

they can be successively removed, and this concentration carried to any extent desired, or until the recovery of the remaining metal is comparatively simple.

2. By treating the matte either solid or liquid with an insoluble molten body having a greater attraction than the matte itself for gold and silver, and thus remove them more or less completely by absorption.

Of the two courses the latter seems to offer distinct advantages for experimental research, inasmuch as the first with various modifications is in constant use at the present time at different reduction works, and its limitations are quite well defined.

To have a possibility of application, the absorbent must be obtainable at a price not too excessive. Its heat of combination with sulphur should be lower than those of iron and copper, lest by reaction it be converted into a sulphide. It must be comparatively insoluble in matte to avoid its excessive loss and possible interference with subsequent treatment of the matte. It should possess great power of absorption so that the gold or silver may be removed as completely as possible.

The following substances may be selected from the elements and their compounds as possessing to a degree qualifications such as to render them tentatively suitable as the absorbent bodies, namely, metallic sulphides, etc., aluminum, lead, copper and iron.

Metallic Sulphides, etc., as Absorbents.

The metallic sulphides, arsenides, antimonides, and perhaps a few rarer compounds of the same general nature, possess properties which render them of peculiar interest in this connection. Those of lead, iron and copper are useful carriers of the precious metals in furnace-practice. Being stable compounds there is not the same tendency of reaction that there is in the case of metallic absorbents in contact with sulphides.

As it is from metallic sulphides that absorption must be made, it follows that to apply the sulphides, etc., as absorbents, there must exist a considerable difference in their attraction for gold and silver. That this exists is shown by the slight affinity between gold and pure ferrous sulphide,[2] whereas in the impure

[2] This section is discussed more fully later on in this paper under the heading "Metallography of Mattes."

state of ordinary matte it is a very powerful absorbent. These differences in absorptive power exist in the antimonides and arsenides as well, and may no doubt be intensified by proper treatment.

The extreme miscibility of the sulphides, and to a certain extent also the solubility of the speisses in each other and in sulphides, is the main factor which prevents the application of these bodies for the purpose of extracting gold and silver from matte. It remains to be seen whether, by proper treatment, the miscibility of the sulphides, etc., cannot be overcome and thus allow their use as absorbents. The field for investigation in this direction is indeed a broad one, and one which promises valuable results.

In an experimental way some half-a-dozen fusions were run upon different sulphides in an effort to obtain a stratification of the matte, but the action of diffusion was too great to allow of such a separation being accomplished with ease.

Aluminum as an Absorbent.

Aluminum and gold combine with avidity to form alloys of the same class as the purple one discovered by Sir William Roberts-Austen, the heat of the combination being calculated by Richards,[3] from an experiment by Roberts-Austen, to be about 1,300 kg-cal. per kg. of aluminum, and on account of this avidity for gold it was hoped that aluminum might prove to be an efficient absorbent from matte. Aluminum, however, possesses a heat of combination with sulphur, to form aluminum sulphide, Al_2S_3, which is far greater than that of either copper or iron. For iron-matte this relation would be as 23,576 g-cal. (the heat of formation of a gram-molecule of ferrous sulphide) is to 124,401 g-cal. (the heat of formation of a gram-molecule of aluminum sulphide), showing the probability of a strongly exothermic reaction between metallic aluminum and ferrous sulphide. In the face of this fact it might yet be possible to heat aluminum up to such a point that it would absorb gold, and yet not sufficiently high to induce the reaction.

Experimentally, this hope was shown to be vain for, at a low red heat, combination takes place readily, and aluminum sheet

[3] Richards' *Aluminium*, p. 502 (1896).

heated in contact with matte to a point slightly lower than that needed for reaction to occur, was not found to contain an appreciable quantity of gold.

The reaction which takes place between metallic aluminum and this matte is mainly confined to the ferrous sulphide present, small quantities only of copper and silicon being reduced. The products are aluminum sulphide and metallic iron containing small amounts of copper and silicon.

The principal reaction is: $3FeS + 2Al = 3Fe + Al_2S_3$.

This reaction takes place when the mass has been heated to the neighborhood of 700° C., and is accompanied by vivid light.

The heat liberated is indicated roughly in the following thermal equation:—

$$3FeS + 2Al,3S = 3Fe,3S + Al_2S_3.$$
$$* (+ 70728) + (- 124401) = (- 70728) + (+ 124401).$$

* These heats of combination are taken from Richards' *Aluminium*, p. 235.

The data given in the above equation correspond to 319.48 g-cal. per gram of iron separated.

The above reactions pre-suppose that aluminum is not added in excess of the ferrous sulphide present, otherwise copper also will be reduced. The iron which is reduced settles as a button to the bottom of the crucible, and the discovery of this reduction of iron from mattes proved to be of value later in the study of iron-bottoms.

Lead as an Absorbent.

The use of lead as an absorbing-agent is probably the oldest method of extracting gold and silver from matte. Processes accomplishing this end are described in all of our older text-books upon metallurgy, but, outside of regular lead-reduction methods, few are in practice to-day on account of the waste of lead incurred, as well as the contamination of all of the products with lead, copper and sulphur.

Matte may be treated with lead for the purpose of extracting the precious metals; by the soaking-process, in which the lead is molten but the matte is not (a representation of which is the Crooke process);[4] by smelting with lead-ores, as in ordinary

[4] Collins, *Metallurgy of Silver*, p. 317 (1900).

practice; or, as has been proposed, by the Davies or Probert process,[5] in which both lead and matte are molten. The lead is introduced into the molten matte-bath as oxide and subsequently reduced, or it may be introduced as granulated metal. In both cases the lead and matte are thoroughly mingled by agitation and allowed to separate by settling.

In a description of Deadwood practice, Collins says:[6] "The matte is worked up by raw smelting with lead ores, as in the ordinary lead-smelting practice. In doing so it is worthy of note that it takes up silver from the charge while giving up nearly all its gold to the lead-bullion."

In order to ascertain the behavior of this matte toward lead, two charges were made containing respectively 10 per cent. and 20 per cent. of lead oxide (lithage) and fused. The fusion was stirred with a wooden stick to ensure thorough mixing and then cast. The results of these fusions are indicated in Table I., experiments Nos. 84 and 86. This operation may be taken as representing the application of the Probert process to the matte. The results yielded are approximately the same as those given by melting the matte upon a bath of lead in a reverberatory furnace, as was done in some experiments at the Deadwood plant.

The results of metallurgical experience indicate that it is not possible to bring lead and matte in contact in such a way as to effect an absorption of the precious metals without also absorbing copper and likewise contaminating the matte with lead to a serious extent.

This is the greatest objection to this type of process, as the percentage of absorption is satisfactory enough, so that the matte could be again charged into the matting-furnace were it not for the loss of lead thereby incurred.

Copper as an Absorbent.

The old Swansea extra process for the production of "Best Selected" copper is the most notable application of copper as an absorbent from matte. This process has been studied by

[5] Eissler, *Metallurgy of Argentiferous Lead*, p. 184 (1891).
[6] Collins, *Metallurgy of Silver*, p. 268.

Mr. Allan Gibb[7] and also by Mr. Edward Keller[8] and a large mass of exact data is at hand representing the deportment of metallic copper toward impurities, including gold and silver, in copper-mattes of high copper-tenor.

Mr. Gibb found in his research that "with a reduction of 8.2 per cent. of the total copper as bottoms, they contain 41.5 per cent. of the total gold; whilst, when 14.4 per cent. of the copper is reduced to bottoms, the whole of the gold is found in them. In some cases traces of gold remained in the regulus; but they were too small to be weighed, even when 50 grams of the samples were taken for assay." "Silver,—silver nearly reaches its maximum concentration in bottoms when 19 per cent. of the total copper is separated in this form, 42.9 per cent. of the total silver being then contained in them." . . . "It is worthy of note that its concentration is lessened by the presence of arsenic and nickel," as shown when the copper is in the bottom is 19.3 per cent., it contains only 26.7 per cent of the total silver. The disturbing factors in this case are shown in the following analyses:—

Analysis of copper-bottom: Cu, 73.31; Ni, 10.34; As, 10.91 per cent.

Analysis of regulus: Cu, 66.56; Ni, 2.52; As, 0.70 per cent.

It must be noted, as is illustrated above, that the influence of other elements, than those under study, and which may be present through intention or otherwise, must be constantly borne in mind and observed in order that the experiments shall be definite and to the point. A slight difference in the method of preparation or manipulation may have as great an effect also, and it is always wise to provide for this contingency by comparing the results of at least two methods of preparation, etc.

Most of the mattes studied contained a percentage of copper approaching that of pure cuprous sulphide (79.82), in which copper is very insoluble. This insolubility does not hold for matte containing a large percentage of ferrous sulphide and in matte containing Fe, 60 per cent., and Cu, 10 per cent., a reac-

[7] The Elimination of Impurities during the Process of Making "Best Selected" Copper. Third Report of Alloys Research Committee, *Institution of Mechanical Engineers*, p. 254 (1895).

[8] Elimination of Impurities from Copper Mattes, *Mineral Industry*, vol. ix., p. 240 (1900), and A Study of the Elimination of Impurities from Copper-Mattes in the Reverberatory and the Converter, *Trans.*, xxviii., 127 (1898).

tion occurs with metallic copper similar to that with aluminum resulting in the separation of metallic iron and the production of copper sulphide. The readiness with which this reaction takes place bars metallic copper from being used as an absorbent for the matte which we have under consideration.

Iron as an Absorbent.

The study of iron-bottoms reduced from these mattes was suggested by a variety of circumstances, chief among which were: 1. The fact that when such ferruginous mattes are produced in conjunction with the treatment of gold- and silver-ores, great masses of metallic iron, or "sows" assaying high in gold, often collect in the blast-furnace bottom. 2. The parallel concentration which is effected by copper-bottoms thrown down, as exemplified in the Welsh process for "Best Selected" copper. This separation in copper-bottoms has been ably studied by Mr. Allan Gibb[9] and also by Edward Keller,[10] and the behavior of gold and silver toward copper-bottoms is fairly well known.

On account of their capacity for absorbing gold, it is very desirable to learn more in regard to the behavior of the iron-sows, which collect in the bottom and forehearth of the matting-furnace.

Dr. F. R. Carpenter[11] says of them: "They were virtually blocks of solid metallic iron weighing from 20 to 40 tons, and carrying sometimes 20 oz. of gold per ton." This is in accordance with my experience while in the laboratory of the Deadwood plant, excepting that, as I remember, the sows usually assayed somewhat higher.

In regard to iron-bottoms it may be of interest to quote Dr. F. R. Carpenter in regard to the practice while he was superintendent of the Deadwood & Delaware Smelter, now the Golden Reward, Consolidated Gold Mining & Milling Co. of Deadwood, So. Dak. He says: "It was my theory that this metallic iron helped to clean the slags, and I did not really care to overcome its formation; but others have tried to do so. It was my theory that if the gold were not recovered in the sows, it would be lost in the slags; and here is, in my opinion, the solution of the question of clean slags in the use of iron-pyrites free from copper."

[9] *Loc. cit.* [10] *Loc. cit.* [11] *Trans.*, xxx., 769 (1900).

Little or no data of an exact or numerical nature seems to have been published upon the function, the relation, and possibilities of this metallic iron in mattes of this character, and it was the original object of this investigation to ascertain as nearly as possible the availability and rate of concentration of gold and silver in iron-bottoms thrown out of a commercial matte of this type.

This portion of the research has been carried out in a manner very nearly parallel to the laboratory-experiments of Mr. Allan Gibb described in the article formerly referred to. This parallelism has been maintained on account of its wide application to cases of absorption-phenomena, and because its clear, direct application renders its deductions easily grasped by a comparatively non-technical mind.

The above renders the results of each research strictly comparable with the other, and it is to be hoped that any future investigations may be capable of a like presentation.

Scheme of Procedure.

Of the five substances tentatively assumed as suitable for use as absorbents, iron seems to be the most inviting one. It is low priced and available in nearly all localities; it possesses a heat of combination with sulphur only slightly greater than that of copper; it is not extremely soluble in ferrous matte, and it possesses a superior affinity for gold. All of which are necessary for its successful application as an absorbing medium, and for purposes of research it presents one very necessary attraction, that is, a lack of data as to its availability and behavior.

With a view of investigating this phase of the question, it was decided to separate the iron contained in a standard matte in various percentages, varying from 10 to 75 per cent. of the total iron. This iron would be collected in a button or so-called "bottom," from the assay of which, together with that of the regulus, the distribution of the gold in the two portions could be readily ascertained.

In order to guard against the chance of being seriously misled by the influence of the precipitating reagents, at least two quite unlike methods of reducing the bottom should be employed.

Actually five methods were used, namely: I. Poling at a

high temperature. II. Reduction by metallic aluminum. III. Reduction by potassium ferrocyanide. IV. Reduction by metallic copper. V. Addition of metallic iron.

The matte upon which these experiments were conducted was furnished by the National Smelting Co. of Rapid City, So. Dak. It is as nearly a duplicate of the matte produced by the well-known Deadwood and Delaware Smelter, now the Golden Reward Consolidated Gold Mining and Milling Co. of Deadwood, as products from different works can be. It is produced under the same conditions from ores of the same district, by the same type of plant, and by men who gained their experience in the original Deadwood works. The object is the same at both works, namely, the recovery of gold and silver from the refractory siliceous ores of the Bald Mountain, Ragged Top and Ruby Basin districts.

These ores contain 80 per cent. or more of silica, and are smelted with a flux of dolomitic limestone, and the addition of sufficient barren pyrite to form a matte of the required tenor. The slag is remarkably high in silica. An approximate analysis of the Deadwood slag, given from memory, is:

SiO_2, 50; FeO, 11; Al_2O_3, 6; CaO, 23; MgO, 10 per cent. Due to the silica-content and the consequent elevation of the fusing-point a very high temperature is necessary in the lower part of the furnace, and a powerful reducing action results. To these conditions is probably due the character of the matte and its remarkable cleaning action on the slag.

Following is the analysis of the matte experimented upon.

Fe, 61.68; Cu, 11.205; S, 25.47; SiO_2, 0.662; Au, 0.0457; Ag, 0.092; Pb, 0.050; Zn, 0.00; Ba, 0.047; As, 0.22; Sb, 0.00; Mn, 0.00; Sn, 0.00; Ca, 0.21; Total, 99.6817 per cent.

I. *Reduction by Heating or Poling.*—Copper-bottoms are obtained from rich, partly roasted, copper-matte by simple fusion, resulting in the precipitating of metallic copper, according to the following reaction:—

$$2Cu_2O + Cu_2S = SO_2 + 6Cu.$$

This result cannot be obtained in the case of a ferrous matte, as was demonstrated by the fusion of varying mixtures of raw and roasted matte.

Charcoal powder was tried, and also poling with a green stick,

to simulate the reducing action of the furnace. In all of these fusions, which were brought to about 1,400° C.,[12] molten matte occupied the lower portion of the crucible, while above floated iron oxide partly slagged by its attack upon the silica of the crucible. The cast of this melt yielded only matte and slag.

Thus the attempt to reduce metallic iron from slightly oxidized matte by poling was not successful, but it was found that a simple fusion of the raw matte at a high temperature yields a nearly constant percentage of bottom, being about 6.5 per cent.

Bottoms designated in Table I. as No. 11 and No. 25 were obtained in this manner.

II. *Reduction by Metallic Aluminum.*—The reduction of iron was attempted by the use of powdered aluminum so as to secure a very intimate contact of the reduced iron with the mass of the matte, but the use of the powdered form was found to be objectionable, and metallic aluminum rods were substituted with success.

The aluminum sulphide produced is difficultly fusible, and if the quantity of aluminum added in the powdered state is sufficient to cast down an iron-bottom, the mass in the crucible is converted into a refractory clinker.

If less than about 5 per cent. of metallic aluminum be added, no iron is cast down, but above this point the bottoms rapidly increase in size until the equivalent of iron is reduced.

The fusion was conducted as follows: The weighed portion of matte was melted in a graphite crucible, and into this the rod of aluminum was plunged, and slowly fed downward with a stirring motion, as it was consumed.

No difficulty was experienced with the fusion, and it remained perfectly fluid after the introduction of as high as 15 per cent. of aluminum. The crucible was allowed to stand for 5 min. and poured into a large conical iron mold. After cooling, the whole was weighed and the iron-bottom broken out, carefully cleaned from the adherent matte, weighed and crushed in a diamond-mortar for sampling.

[12] On account of the extremely corrosive action of these iron-mattes upon the protecting tube of the thermo-couple, the exact temperature was not measured, but a close approximation was secured by allowing the thermo-junction and tube to rest just above the surface of the matte for a few moments. Even under these precautions, the spray constantly thrown off proved to be very destructive.

The residue, composed of sulphides of copper, aluminum, iron, etc., very rapidly disintegrated in the air with the evolution of hydrogen sulphides (H_2S), due to the reaction of aluminum sulphide upon atmospheric moisture, as follows:—

$$Al_2S_3 + 3H_2O = Al_2O_3 + 3H_2S.$$

To avoid change of weight the samples were assayed immediately.

Aluminum sulphide does not readily mix with the other sulphides of the matte, and is often found at the top of the cast as a distinct, lighter layer, assaying low in gold and silver.

III. *Reduction by Means of Potassium Ferrocyanide.*—As the result of a number of experiments, it was found that if a charge of finely ground matte be mixed with ground potassium ferrocyanide and fused, the following changes will take place progressively:

1. Gradual decomposition of the ferrocyanide takes place with the evolution of ammonia fumes, due to the reaction of the nitrogen of the ferrocyanide with its water of crystallization.

2. A process of liquation takes place, resulting in a porous crust floating on molten sulphides.

3. On further heating or stirring, this crust dissolves in the sulphide, and on pouring, a bottom or metallic portion is rarely found.

4. If this crust be separated in the second stage by pouring and skimming-back the crust, and then melt it down alone, it will result in the major portion separating as a metallic bottom covered with a thin layer of sulphide.

In this manner were prepared the bottoms in Group II.

It is possible thus to prepare almost any desired percentage of bottom, but the percentage will depend not only upon the quantity of ferrocyanide used, but also upon the temperature employed, and the time of heating after liquation, and before skimming. Long heating and high temperature tend to produce small bottoms.

The mattes after treatment are greatly changed. The fracture becomes columnar and highly crystalline, with broad cleavage-surfaces. This change is due to the introduction of the

alkali metals, in this case potassium, doubtless as the sulphide. This is demonstrated by melting matte with potassium sulphide, when perfect intermixture takes place with the above-noted change in fracture. It is worthy of note that pure copper sulphide shows the same deportment with potassium sulphide.

A very rapid oxidation of these mattes takes place upon exposure to air, with consequent disintegration. This oxidation is so rapid that freshly ground matte will rapidly attain a temperature of 60° C. if exposed to air. It is greatly hastened with proportionate increase in temperature by directing a gentle stream of oxygen upon the matte.

In experiment No. 81, lead oxide (PbO), equal to 10 per cent. of the matte-charge, was added in addition to the ferrocyanide, in order to observe the influence of lead-bearing mattes upon the absorption of gold and silver in the iron-bottom. The fusion was conducted without skimming, and an iron-bottom was found in the cast, free from any accompanying reduced lead. It will be seen upon referring to Table I. that this addition of lead oxide which combined to form matte is practically without influence upon the absorption.

The original matte undergoes a liquation with the separation of a crust similar to that just described, but the crust in this case does not yield a metallic bottom. Three of these crusts were separated by skimming, and as they give results almost identical they are represented by a single sample, designated as No. 61 in Table I.

IV. *Reduction by Metallic Copper.*—The addition of copper was made with the intention of obtaining copper-bottoms. The charge for experiments Nos. 68 and 70 was made up of finely ground matte, to which was added 30 per cent. of the weight of fine shot-copper and a liberal excess of charcoal-dust to prevent oxidation. The casts from these fusions revealed upon examination not a copper-bottom, but an iron-bottom which clung tenaciously to the matte.

The second charge, experiment No. 90, was made up in the same proportions as Nos. 68 and 70, but omitting the charcoal-dust. The casts resulting did not show the separation of either metal as a bottom. However, the fracture of the regulus from

these fusions showed small crystals of iron in great abundance. (Fig. 6 is a microphotograph of a polished section of No. 69.)

V. *Reduction by Metallic Iron.*—Fine iron-drillings mixed with the ground matte and fused yield a bottom of iron.

The incomplete contact with the matte, as the iron sinks during fusion, would lead one to expect the low percentage of extraction, which is indicated by the line representing the results with those irons on Figs. 1 and 2, Group III.

EXTRACTION OF GOLD AND SILVER FROM IRON BY MEANS OF LEAD.

Metallic iron weighing 1,531 g., assaying 17.21 oz. gold and 5.2 oz. silver per ton, was melted in a graphite crucible, and 794 g. of test-lead was added in portions and thoroughly stirred. The resulting lead-bottom weighed 466 g. and the residual iron weighed 1,802 g., indicating that it had taken up about 271 g. of lead. The assays showed that the gold and silver had been largely absorbed by the lead, and the results designated as experiment No. 96 show the percentage of extraction. It will be seen that the transfer of gold and silver to the lead is fairly complete, but the retention of so much lead by the iron would be a serious factor against its treatment by this method.

In obtaining the following results, great care has been exercised to use accurate methods of estimation.

For the assay of the iron-bottoms the combination method was used. The sample was dissolved in dilute sulphuric acid, lead acetate and a slight excess of salt solution was added and the precipitate and residue filtered off. The dried filter-paper and contents were then scorified and cupelled as usual. This method checked against the direct scorification with test-lead and litharge, and was preferred for the reason that a larger sample could be taken.

For the assay of the regulus and mattes, the crucible method was employed, using a heavy charge of litharge as recommended W. G. Perkins.[18]

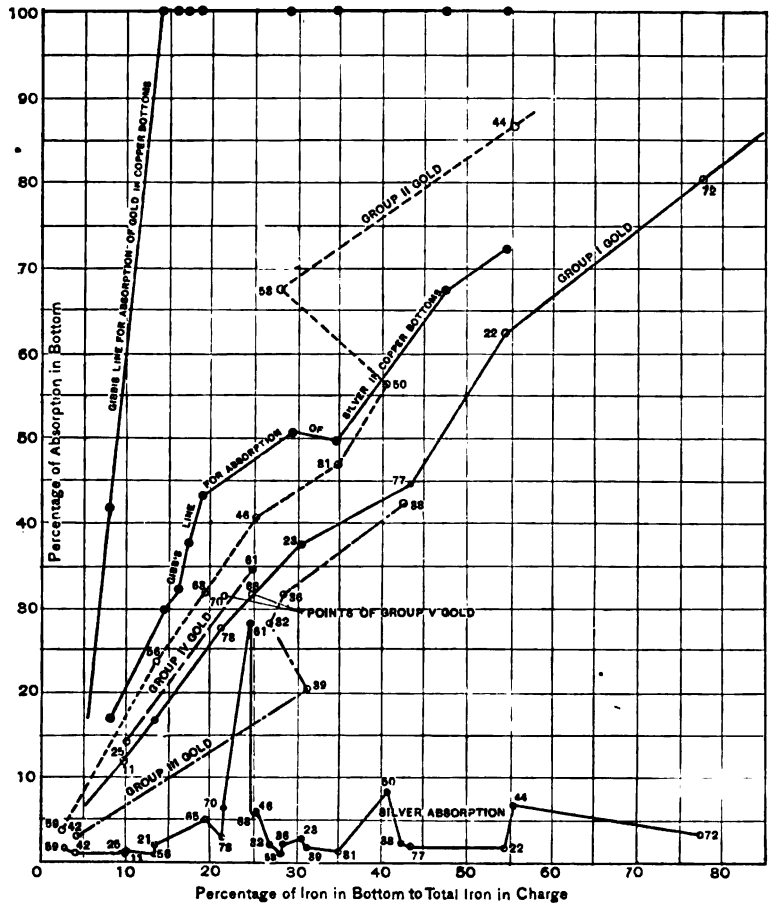
The results have been classified in groups according to the method of separating the iron-bottoms. The percentages of bottom has been calculated:—1. Directly as percentages of the

[18] The Litharge Process for Assaying Copper Products, *Trans.*, xxxi., 913.

total weight of the fusion; that is, the weight of bottom plus the weight of regulus; as is shown by the following:

$$\frac{\text{Weight of Bottom} \times 100}{\text{Weight of Regulus} + \text{Weight of Bottom}} = \text{Percentage.}$$

FIG. 1.



2. As percentages of iron in bottom to total iron in regulus and bottom.

This is more logical, as it thus makes the percentage of absorption a direct function of the percentage of iron thrown down. This method was followed by Allan Gibb[14] in his study

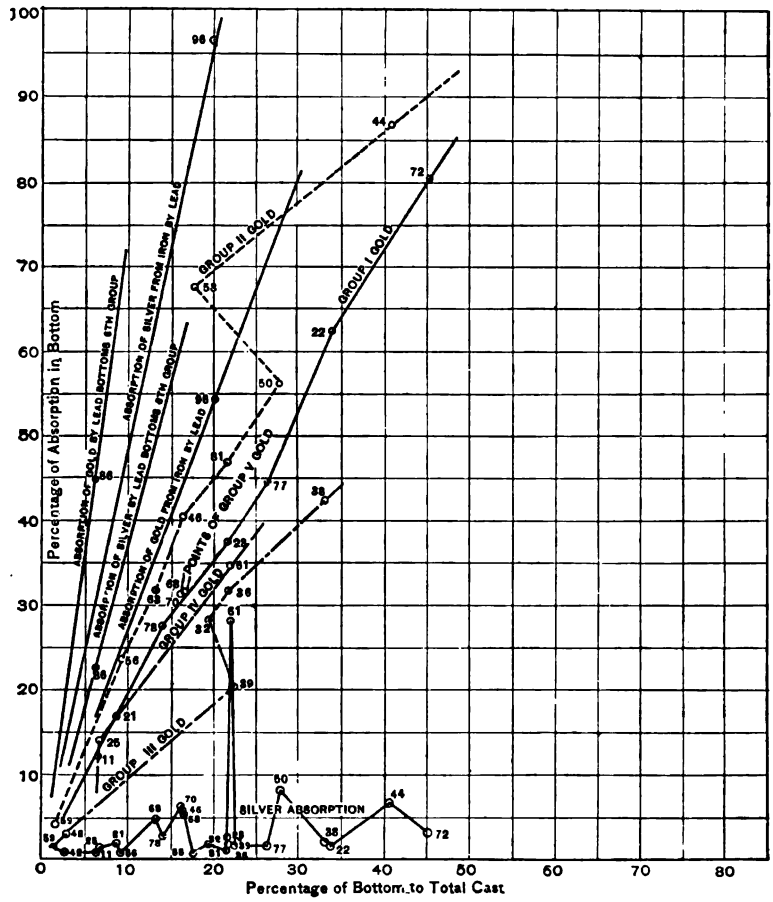
[14] The Elimination of Impurities during the Process of Making Best Selected Copper. Fourth Report of the Alloys Research Committee, *Institution of Mechanical Engineers*, p. 254 (1895).

of copper-bottoms, and his results can be compared directly with those herewith given.

The method of computation is shown by the following form:

$$\frac{\text{Weight of Bottom} \times \text{Per Cent. of Fe in Bottom} \times 100}{\text{Weight of Regulus} \times \text{Per Cent. of Fe in Regulus} + \text{Weight of Bottom}} = \text{Percentage,} \times \text{Per Cent. of Fe in Bottom.}$$

FIG. 2.



The results are plotted in Cartesian co-ordinates on Figs. 1 and 2. The percentages of gold and silver respectively contained in the bottom being placed as abscissa, and the percentage of bottom as ordinates.

These points have been connected by straight lines in order

to facilitate inspection, and to render at once evident to the eye, the relations which the different groups bear to each other. The grouping of the points representing silver-absorption was not attempted on account of the low and variable values obtained.

The experiment number is attached to each point plotted to render reference to the figures and tables easily made.

The lines representing absorption by lead are drawn in their probable position, but this does not constitute an attempt to fix them in this position, as enough determinations were not made to render this point rigidly exact.

The curves obtained for silver and gold, by Mr. Allan Gibb in the research previously referred to, have been introduced for comparison on Fig. 1, in which the values obtained for gold or silver in iron have been similarly derived.

On comparing the results obtained by the different methods of precipitating iron-bottoms (Table I.), it will be seen that there is not as great a divergence as might have been expected considering the variety of reagents employed. It is seen in Fig. 1 that the points fall roughly into three sets:—1. Showing the greatest absorption, the iron-bottoms thrown down by means of potassium ferrocyanide. 2. Showing the least absorption, namely, those bottoms precipitated from mattes by addition of metallic iron. 3. The indefinite, intermediate one comprising the bottoms thrown out by metallic copper, the bottoms thrown out by high temperature, liquation or poling, and those thrown out by metallic aluminum.

From the position of the line representing the aluminum-iron bottoms it appears that the presence of metallic aluminum does not facilitate the collection of the precious metals as might have been expected from the energy of combination which aluminum and gold exhibit. This may in part be due to the very complete reaction which occurs between aluminum and the sulphides present.

Most marked of all perhaps is the contrast (shown in Figs. 1 and 2) between the absorption-percentages of gold and silver.

It is seen that while gold is carried down in a moderate degree by iron, silver is only slightly so; indeed it is very likely that the major portion of the silver carried down was not as an alloy with iron, but included with globules of matte in the bottom being an emulsion, as it were, of matte in metallic iron.

TABLE I.—*Results Obtained by Various Methods of Precipitation.*

Experiment No.	Weight of Regulus.	Weight of Bottom.	Quantity Cast as Bottom.	Iron.		Total Iron in Bottom.	Regulus Contains.		Bottom Contains.		Total Gold Separated in Bottom.	Total Silver Separated in Bottom.
				In Regulus.	In Bottom.		Gold.	Silver.	Gold.	Silver.		
	GROUP I. Bottoms separated by metallic aluminum.											
	Grams.	Grams.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Mg.	Mg.	Mg.	Mg.	Per Cent.	Per Cent.
21	746	72	8.8	59.1	93.6	13.3	336	810	68.5	16.0	16.9	1.9
22	637	324	33.7	41.2	95.7	54.2	174	853	285.0	12.1	62.3	1.4
23	710	197	21.7	59.3	94.1	30.6	277	666	166.0	22.2	37.4	2.5
72	582	475	45.0	23.7	96.1	77.0	88	877	358.0	27.7	80.3	3.0
77	718	254	26.2	44.6	94.7	43.2	253	882	204.5	13.6	44.6	1.5
78	884	145	14.1	58.6	95.3	21.1	335	880	126.8	23.4	27.5	2.7
	GROUP II. Bottoms separated by potassium ferrocyanide.											
44	628	480	40.6	52.4	93.8	55.2	68.5	791	446.0	55.8	86.8	6.6
46	750	147	16.4	53.6	91.6	25.1	258.3	787	174.5	48.5	40.4	5.8
50	810	312	27.8	53.7	94.3	40.4	221.7	853	285.6	74.0	56.1	8.0
53	819	176	17.7	51.4	92.1	28.0	153.0	895	282.0	7.8	67.2	0.7
68	810	124	13.3	58.2	90.8	19.3	306.0	876	144.0	45.2	31.9	4.9
56	895	92	9.3	61.4	91.2	13.3	344.0	880	105.0	7.0	23.4	0.79
59	947	16	1.66	60.8	89.1	2.4	420.0	885	15.5	12.0	3.6	1.3
81 (*a*)	819	226	21.6	45.6	88.0	34.8	237.5	922	206.5	9.3	46.8	1.0
	GROUP III. Bottoms separated by addition of metallic iron.											
32	930	226	19.5	63.0	95.2	26.8	319	900	124.5	16.5	28.1	1.8
36	927	259	21.8	66.0	93.0	28.3	297	834	137.0	16.2	31.6	1.9
38	830	407	33.0	63.2	94.3	42.2	259	807	188.5	16.8	42.2	2.0
39	850	247	22.5	63.1	97.1	31.0	351	892	90.0	13.6	20.2	1.5
42	854	24	2.74	65.8	96.8	4.0	449	888	14.2	5.9	3.07	0.7
	GROUP IV. Bottoms separated by high temperature or poling or liquation.											
11	607	44	6.5	62.0	92.7	9.8	244	554	33.6	4.5	12.1	0.8
25	1200	86	6.7	62.2	96.5	10.0	482	940	79.3	10.3	14.1	1.1
61 (*b*)	736	207	22.0	62.0	72.8	24.8	298	646	155.4	251.0	34.4	28.0
	GROUP V. Bottoms separated by addition of metallic copper.											
68	1068	212	16.6	52.2	86.2	24.7	303	864	140	47.5	31.6	5.2
70	1062	205	16.2	53.1	89.5	21.5	310	842	140	54.7	31.2	6.1
	GROUP VI. Lead-bottoms thrown out of matte.											
84	1072	11.47	1.06	.....	.....	.....	410	883	23.24	29.26	5.36	3.2
86	1053	67.85	6.05	53.9	.....	.....	248	698	201.96	201.3	44.9	22.4
	F grams.	Pb grams.										
96	1802	466	20.1	.....	.....	.....	409	12.4	483.37	257.7	54.2	95.4

(*a*) In the presence of lead-matte.
(*b*) Group VII. Liquation of matte into crust or regulus.

This also is sufficient to explain the irregularity exhibited in the values for silver-absorption. The relation of gold and silver in the bottoms agrees very well with the results observed in furnace-practice where iron "sows" are produced.

Experiment No. 61 is an exception to this rule, as it is the crust skimmed off from a fusion of raw matte and charcoal. It is mainly composed of the crystals of iron, shown in the metallographic section, from which the sulphides have been liquated.

The percentage of iron (Table I., Group III., IV. and V.)

has been changed during the process of liquation, so that after the operation the crusts contain 72.8 per cent. of iron and the liquated portion only 62 per cent.

The accompanying change in the distribution of the gold was such that 34.4 per cent. of the gold is concentrated in 22 per cent. of crust.

Referring to Fig. 2 it will be seen that the absorption of gold in iron-bottoms corresponds quite regularly (but is somewhat lower) to the absorption of silver in copper-bottoms as found by Mr. Allan Gibb.

Metallic iron is easily soluble in the mattes worked with, as is shown in the microscopic examination given later in this paper under the heading "Metallography of Mattes." The solubility increases with the temperature, and at these temperatures employed for the reduction of bottoms it was very marked. The attack of matte upon the bottoms produced a rough, dirty surface, which was very difficult to clean properly.

This re-solution would if long continued be very likely to alter the percentage of gold contained in the bottom. It being analogous to scorification, with consequent concentration. It is likely that to this action can be ascribed the irregularity of the points plotted in the absorption-values.

This view is upheld by the regularity of the results obtained in the Group I.; those buttoms precipitated by metallic aluminum, as these buttons were notably smooth and unattacked.

Conclusions.—1. Gold is collected in iron-bottoms in a ratio somewhat less than the collection of silver by copper-bottoms.

2. Not more than 5 per cent. of the silver can be expected to enter the iron-bottoms.

3. The precious metals tend to collect in the crust produced in liquation of the matte.

4. Provided other circumstances allow the use of metallic iron as a carrier, gold may be expected to be satisfactorily separated from the other products of fusion, but silver will not be satisfactorily collected.

5. Copper-bottoms cannot be obtained from mattes of this character on account of the reaction of copper with ferrous sulphide. Experiments Nos. 68, 70 and 90, point to this conclusion.

6. Aluminum does not markedly influence the absorptive

power of iron-bottoms. Shown by results of first group of bottoms.

Metallography of Mattes.

The chemical composition of mattes is subject to very great variations, and hence is not capable of a general quantitative expression, excepting through such wide limits that it becomes next to meaningless.

Edward Keller[15] groups the various elements entering into its composition about sulphur as a nucleus, and regards the quantity of this element as nearly a constant placing the limits at 22 and 24 per cent.

Herbert Lang[16] goes even further and classifies the arsenides and antimonides, known as speisses, along with the mattes under one family and designates them as sulphide-mattes, arsenide-mattes, and antimonide-mattes.

The excessive variation of which matte is capable will be illustrated in Table V. taken from Herbert Lang.[17] These data illustrate the extreme variation in percentage of the elements named, as shown by the analysis of a number of widely varying types of mattes.

Table V.— *Variation in Composition of Mattes.*

Component.	Highest.	Lowest.	Component.	Highest.	Lowest.	Component.	Highest.	Lowest.
	Per Ct.	Per Ct.		Per Ct.	Per Ct.		Per Ct.	Per Ct.
Iron.....	70.47	0.136	Manganese....	3.00	0.000	Calcium ...	7.00	0.000
Copper.	80.00	0.000	Silver.....	5.00	0.000	Barium.....	22.00	0.000
Lead....	73.00	0.000	Gold.....	0.11	0.000	Sulphur....	44.00	trace
Zinc.....	11.50	0.000	Platinum.....	0.008	0.000	Arsenic.....	52.00	0.000
Nickel..	55.00	0.000	Bismuth.....	1.26	0.000	Antimony.	60.00	0.000
Cobalt..	54.00	0.000	Molybdenum.	2.31	0.000			

The profoundly complex nature of this metallurgical product is rendered evident by the number of elements shown in Table V. which may enter into its composition.

The constitution, meaning by this term, the compounds actually present and their several physical states, is even more complex.

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[15] *Mineral Industry*, vol. ix., p. 242 (1900).

[16] *Matte Smelting* (1896).

[17] *Matte Smelting* (1896).

Keller[18] has shown that iron may exist as the magnetic oxide, and Münster[19] and Farbaky[20] advance the idea that metallic iron exists in solution and separates out on cooling.

This existence of metallic iron and its magnetic oxide, as well as various other compounds which have been isolated, would seem to point to a composite structure in matte. That this point is not altogether clear is rendered evident by the following question, in regard to constitution of mattes, taken from Peters,[21] "Is it a chemical combination, a mixture or a partial alloy?" Upon this same subject Edward Keller[22] says, "The true chemical character of copper-mattes does not yet seem to have been definitely determined, *i.e.*, it is still more or less of an open question whether they are true chemical compounds, or mixtures such as igneous rocks."

With the idea of determining, if possible, some of the conditions existing in the internal structure of matte, a number of specimens were prepared and examined by the methods which have been so fruitful in the field of metallography.

A number of specimens also were specially prepared from the mattes made during the investigation of the absorption by iron-bottoms.

The apparatus employed both for the microscopic and photographic work is well described by W. Campbell,[23] to whom I wish to take this opportunity to express my thanks for his skilful preparation of several of the photographs given herewith.

Fig. 3 shows an unetched section of the original matte just as it was received from the works. In this photograph the white areas represent the iron, the dark areas are cavities and the ground-mass ferrous sulphide. Throughout this ground-mass of ferrous sulphide are distributed numerous slightly lighter colored bodies of rounded contour and smooth solid appearance. These are areas of copper sulphide (Cu_2S).

The effect under the microscope is that of a wavy striated bronze-colored ground-mass in which are set the bright white

[18] *Engineering and Mining Journal*, November 16, 1895, p. 465.

[19] *Berg- und Huettenmaennische Zeitung*, p. 195 (1877).

[20] *Idem.*, p. 184 (1894).

[21] *Modern Copper Smelting*, 7th Ed., p. 230 (1895).

[22] *Mineral Industry*, vol. ix., p. 240 (1900).

[23] Upon the Structure of Metals and Binary Alloys, *Journal of the Franklin Institute*, July and September, 1902, pp. 131 and 201.

crystals of iron and the azure-blue areas of the copper sulphide. This matte was chilled on water-cooled pipes at the works, and therefore the crystallization is not as complete as it would have been in a more slowly cooled specimen.

The general deportment and appearance of the constituents is sufficient to identify them, but as an additional safeguard the following tests were applied. 1. Nitric acid (2 per cent.) rapidly attacked the crystals of iron and blackened them. The ferrous sulphide was but slowly etched and the copper sulphide not at all. 2. A dilute solution of copper sulphate was applied, and the iron was immediately covered with a coating of deposited copper, the ferrous sulphide being slowly coated by copper. 3. Sulphuric acid (1:6) etched both the iron and the ferrous sulphide, but the copper sulphide remained unattacked.

Figures 4 and 5 show similarly prepared sections of mattes obtained during the operation of obtaining iron-bottoms, Fig. 4 being the matte from which bottom No. 59 was obtained by the use of potassium ferrocyanide; and Fig. 5 represents the matte remaining after vigorously poling the original matte at a temperature of about 1,400° C. to obtain bottom No. 25.

The forms of the crystals of iron in these two mattes are very different and indicate a difference in the system of crystallization. This may be due to the difference in temperature to which the two mattes were heated. In obtaining bottom No. 59, the temperature did not rise above 1,150° C. It is, however, to be noted that the cubic and hexagonal sections of the crystals have been observed in the same section of matte.

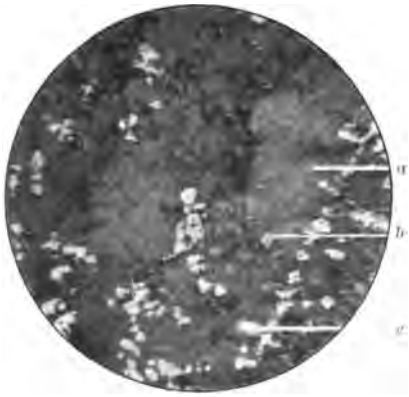
The copper sulphide in Fig. 5 was so dark a blue that it is now shown as a darker constituent in the photograph. It must not be confused with the black cavities which are present.

(The wavy lines in Fig. 5 are due to the Newton's-rings effect caused by a film of oil very difficult to remove.)

An increase of the copper sulphide constituent is seen in Fig. 6. This matte is that from which bottom No. 68 was obtained by addition of metallic copper. The copper sulphide now assumes the rôle of ground-mass and the ferrous sulphide is shown as dark areas.

In Fig. 7 is shown the photograph of a pure copper sulphide etched with strong nitric acid until its crystal cleavage shows in relief. The round black spots are cavities—the "blowholes"

FIG. 3.



a. Iron sulphide. b. Copper sulphide.
c. Metallic iron.
ORIGINAL MATTE.
Magnified 130 Diameters.

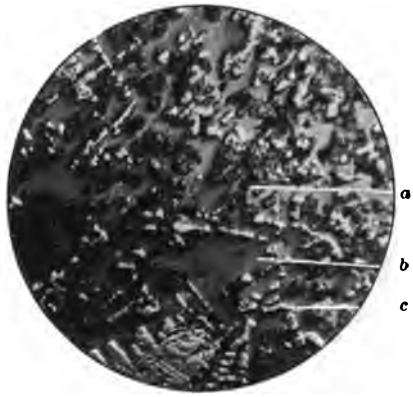
FIG. 4.

No. 59.
TREATED MATTE.
Magnified 60 Diameters.

FIG. 5.

No. 25.
a. Copper sulphide.
TREATED MATTE.
Magnified 120 Diameters.

FIG. 6.



No. 68.
a. Ferrous sulphide. b. Copper sulphide.
c. Metallic iron.
TREATED MATTE (41.2 per cent. copper).
Magnified 40 Diameters.

FIG. 7.

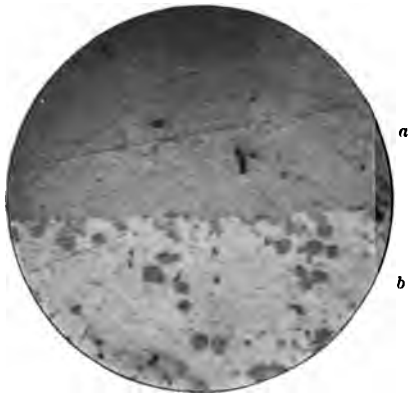
COPPER SULPHIDE.
Magnified 155 Diameters.

FIG. 8.

COPPER SULPHIDE.
Magnified 175 Diameters.

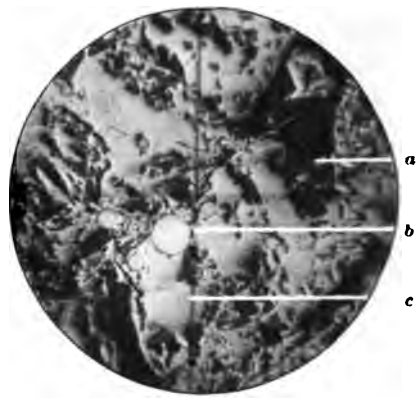
[To face p. 688]

FIG. 9.



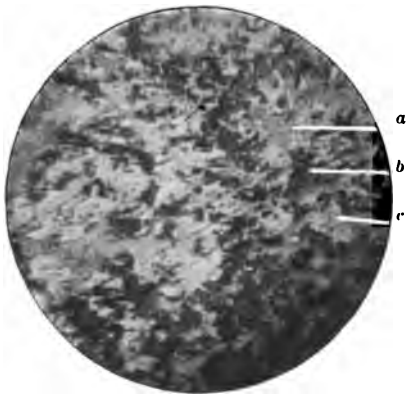
a. Matte. *b*. Copper.
SEPARATION OF MATTE AND COPPER.
Magnified 130 Diameters.

FIG. 10.



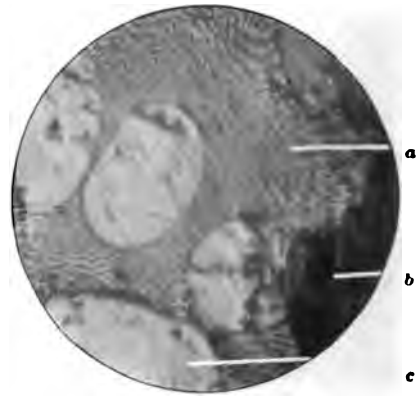
a. Cavity. *b*. Gold. *c*. Sulphide.
SEPARATION OF GOLD.
Magnified 120 Diameters.

FIG. 11.



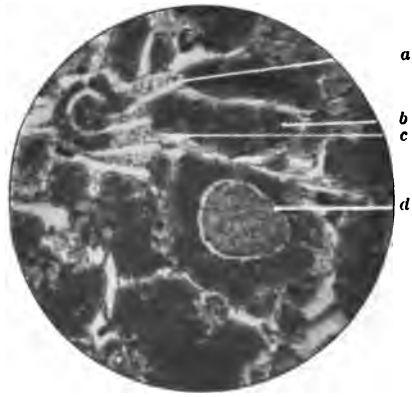
a. Eutectic. *b*. Iron. *c*. Sulphide.
EXCESS OF IRON.
Magnified 270 Diameters.

FIG. 12.



a. Eutectic. *b*. Iron. *c*. Sulphide.
EUTECTIC.
Magnified 780 Diameters.

FIG. 13.



a. Rod-like Crystal. *b*. Eutectic. *c*. Ferrite. *d*. Matte Globule.
IRON BOTTOM—NO. 39.
Magnified 130 Diameters.

of steel parlance. In Fig. 8 is shown a like photograph, except that the pure sulphide has been fused with an excess of sulphur at a low heat. The analysis of the sulphide now is as follows: Cu, 78.86; S, 20.86; SiO_2 and graphite, 0.2; Total 99.92 per cent. The theoretical composition is: Cu, 79.82; S, 20.18; Total 100 per cent.

This matte upon cooling was found to be surprisingly brittle so that large pieces could be crushed by the fingers. Its fracture was unusual, being along the surfaces of large elongated grains having polished black surfaces like those of gunpowder kernels. Evidently there is an excess of sulphur to be accounted for.

A metallographic inspection shows the excess of sulphur, probably in the form of CuS, lying between the crystal boundaries, and thus producing planes of weakness. One of these boundary lines is shown crossing the center of the photograph and it shows how complete is the isolation of each crystal of cuprous sulphide.

The very great insolubility of metallic copper in cuprous sulphide was shown by the analysis of the sulphide portion (upper) of a matte of pure Cu_2S fused in the presence of metallic copper, which was Cu, 79.71; S, 20.1 per cent., which approximates the theoretical percentages within ordinary experimental error.

Figure 9 shows the line of junction between metallic copper and the sulphide, and also the globules of sulphide caught at the moment of solidification as they were rising through the metallic copper to the sulphide above.

Metallic Iron in Matte.

The general impression gained from most writers upon this subject is that the iron occurring in mattes is present as included particles, held in suspension by the viscosity of the medium. That this is not true is seen from the delicate crystallization-forms that the iron exhibits in the foregoing photographs. The iron must be in solution in some manner in the molten matte and separate upon cooling from this solution; perhaps by the decomposition of a lower sulphide.

Whatever may be its state in the molten matte, its presence is most important wherever iron-matte is to be used as a collector of the precious metals.

To render mattes efficient carriers it is the custom of smelters

to provide for a certain percentage of copper, even though the source of the copper be such as to make its treatment uneconomical.

Often circumstances are such as to render it imperative that the copper-content be kept as low as possible, and many attempts have been made to replace it entirely by iron sulphide.

This more or less complete replacement of copper by iron has been found successful[24] up to a certain limit, which in practice may be taken as nearly a 5-per cent. copper-matte. The limit, however, is dependent upon the acidity of the slag produced, being lower in proportion as the slag is acid. But much below this limit varying results are found; some reports being good, and some otherwise.[25]

H. F. Collins[26] says, "The general consensus of metallurgical opinion goes to show that a pure iron-matte, FeS, is an extremely poor medium for collecting gold or silver, especially the former, though an iron-matte containing comparatively small quantities only of Cu, Bi, Fe, and As may be very efficacious when the accompanying slags are of bi- or sesqui-silicate type.[27] With slags of mono-silicate type, and with those still more basic, it seems impossible to successfully concentrate the precious metals without a considerable proportion of copper or lead in the matte."[28]

The high silica and lime-content of the slags advocated by Carpenter,[29] Lang,[30] and Collins[31] tend to a high temperature within the furnace, and this temperature is shown by Le Chatelier and M. Ziegler[32] to facilitate the production of metallic iron in ferrous sulphide.

May not this be the point about which centers the solution of the conflict of reports as to the efficiency of iron-matte as a collector of the precious metals?

Spilsbury's[33] experiments upon reverberatory matting, and the experiments of Pearce[34] show the inefficiency of pure fer-

[24] F. R. Carpenter, *Mineral Industry*, vol. ix., p. 696 (1900).
[25] W. L. Austin, *Trans.*, xvi., 262–268.
[26] *Metallurgy of Silver*, p. 254 (1900).
[27] Herbert Lang, *Matte Smelting*, p. 22.
[28] Pearce, *Trans.*, xviii., 454, 457.
[29] Pyritic Smelting in the Black Hills, *Trans.*, xxx., 774.
[30] *Matte Smelting*, p. 39.
[31] *Metallurgy of Silver.*
[32] *Metallographist*, January, 1903, p. 26.
[33] *Trans.*, xv., 767.
[34] *Trans.*, xviii., 447.

rous sulphide as a carrier for gold, but it is the record of Dr. Carpenter[35] to have obtained a very efficient action from iron-matte.

To study the action of metallic iron in mattes the following experiment was performed: An alloy of gold and pure iron was prepared by melting fine gold and pure soft iron-wire in such proportions as to yield an alloy containing 15 per cent. of gold. The micrographic section of this alloy showed it to be mainly a solid solution of gold in iron, with only small areas of a rich gold-alloy showing between the grains of ferrite.

This alloy was next treated at a red heat with sulphur and converted thereby into ferrous sulphide. The bronze-colored sulphide was non-magnetic, and showed a button of separated gold on its lower surface. Fig. 10 shows a section of this sulphide and the globules of gold which are distributed throughout it. Ferrous sulphide is very difficult to polish, being cracked and filled with numerous cavities due to changes of dimension in cooling and to the presence of absorbed gases. The dark areas are these cavities and cracks, the lighter areas are the ferrous sulphide, which contains the gold as the sharply defined round spots shown in the photograph. This separation of gold agrees with the statement of Pearce referred to earlier in this discussion.

A portion of the ferrous sulphide was ground to an impalpable powder, passed through a 100-mesh screen and vanned to separate as much gold as possible. The purified portion was then ground with mercury in a 2-per cent. solution of potassium cyanide to facilitate amalgamation, and finally washed by treating several times with 1 oz. of pure mercury. The residue was heated to drive off any residual mercury, and the percentage of gold estimated by assay. It still retained 1.044 per cent. of gold, which indicates that gold is not wholly insoluble in ferrous sulphide; but to prove this result conclusively would require a long and tedious operation to render it certain that no gold remained in an extremely finely divided condition.

The ferrous sulphide, together with all the gold, was fused with the addition of sufficient soft iron-wire to bring the re-

[35] *Trans.*, xxx., 768.

sultant button up to Fe, 68 per cent. Actually the button contained Fe, 67.8 per cent. by analysis. This button containing an excess of iron showed no trace of a separation of gold, though it contained 6.288 per cent. of gold as shown by assay.

Two sections of this button are shown in Figs. 11 and 12, taken from the specimen after etching with alcoholic nitric acid of 2-per cent. strength. Three constituents appear to be present: 1, the metallic iron shows as black areas; 2, the ferrous sulphide shown as the light areas; and 3, a gray constituent which is shown under high power in Fig. 12 as the eutectic. In Fig. 12 the rounded areas are ferrous sulphide surrounded by the eutectic, and to one side is seen a body of iron as a bluish area. The constitution of this eutectic is not known. Its two components deport themselves with reagents as though they were iron and sulphide respectively, but, before etching, the dark component shows as a blue-gray, while the bodies of iron to which it seems allied appear as a bluish-white. A similar eutectic, found in ferrous sulphide by Le Chatelier[36] and M. Ziegler, is believed by them to be composed of iron oxide and iron sulphide.

The gold is contained in the iron present and in the dark portions of the eutectic, as is proved by etching deeply with alcoholic nitric acid and then heating strongly, when the golden color of gold replaces the dark areas of iron and the eutectic.

It is thus conclusively proved that though ferrous sulphide in the pure state does not exercise much solvent power upon gold, the presence of an excess of iron converts it into a substance of entirely different deportment towards gold. In this light it seems very probable that herein lies the explanation of the oft-times seeming erratic behavior of iron-mattes. The view which Carpenter takes of the functions of metallic iron in the furnace has been previously quoted, and certainly seems to be upheld by the facts.

Figure 13 shows a section of a typical iron-bottom. It was etched with 2-per cent. alcoholic nitric acid and shows four constituents: 1. Round or oval globules of matte included in the bottom as an emulsion. 2. Ferrite shown as the dark grains surrounded by a network. 3. A dark component of

[36] *Metallographist*, January, 1903, p. 26.

the network having a eutectic structure. 4. Bright, rod-like crystals of unknown nature, but resembling cementite. These crystals are not etched with nitric acid of 2-per cent. strength, and they are not coated with copper upon treatment with copper sulphate. The gold contained in this bottom is so small that efforts to detect its presence as compounds have been without success.

The question of the existence of double sulphides of copper and iron and also the mutual solubility of each in the other under varying conditions of temperature is now being investigated by a research.

It may be said that the evidence now available does not seem to favor the assumption of the combination of iron or copper sulphides to form double sulphides except, perhaps, in very small quantity.

Conclusions.

At present the following provisional conclusions seem justifiable:

1. Mattes are not homogeneous, but are composed of distinct mineral entities, and may be likened to igneous rocks in which the mineral constituents fall out upon the lowering of the temperature and ensuing solidification.

2. The avidity of iron-mattes may be increased by increasing the reducing power and temperature of the furnace, thus obtaining a greater excess of iron in the matte produced.

3. The absorptive power for gold of a matte composed wholly of iron and sulphur is greatly dependent upon the excess of iron contained above that necessary to form ferrous sulphide (FeS).

Notes on the Gold District of Canutillo, Chile, S. A.

BY SYDNEY H. LORAM, CANUTILLO, CHILE, S. A.

(Atlantic City Meeting, February, 1904.)

THE following notes have been compiled more for the reason that the district is little known to the outside world for its gold production, than for the hope of giving valuable information.

The district of Canutillo lies about 15 miles south of the city of Freirina (lat. 28° 50′ S. long. 70° 75′ W.), and contains nine principal veins, all within a radius of a mile; 50 or more claims have been located on these veins, and this paper refers chiefly to those belonging to the Anglo-Chilean Exploration Co., Ltd., on which the greater part of the development work has been done.

In all probability the mines were first worked by the Incas, who, under Tupac Yupangui, captured the country as far south as the Mauli river, between the years 1430 and 1470, and imposed a yearly tribute of gold and copper, which was forwarded by carriers on foot to Cuzco, the Inca capital in Peru. In the year 1535, soon after the time of the Spanish conquest, the stories of these consignments, together with those of supposed precious stones, led Diego de Almargro with a band of 400 soldiers to explore as far south as the Incas had gone, in the hope of finding the source of the precious metal, which he believed to have come from some definite point instead of being, as was afterwards proved, the collection of innumerable districts scattered over the entire length of the country.

The method of winning the gold used by the Incas or by the natives whom they forced to work seems to have consisted in removing the ore by hard-wood wedges driven by stone-hammers, while a human shoulder-blade worn along its rounded edge, found in one of the mines, had evidently done duty to its second owner as a scraper or shovel. The mining was done by steep inclined workings sunk on the lodes at about 40° from the horizontal, the direction being reversed every 10 or

20 m., thus forming zigzags which have rough steps cut in them, but are generally so small that a large man has difficulty in getting through, leaving it somewhat of a mystery as to the manner in which the ore was carried out, except upon the assumption that the raw-hide bags, or *capachos*, used at present must have been of smaller size and carried by children. Ore of sufficient richness was removed entirely, excepting small pillars of from 6 to 8 in. in diameter, which were left at irregular intervals as a support for the walls.

The ore was milled, in what were called *marays*, by rocking a large stone with a slightly convex under surface, back and forth in the hollow of a nearly flat stone underneath, the movement being imparted to the upper stone by long cross-handles lashed to its top. Water was used to assist in removing the crushed ore which was carried away as tailings, while the residue left in the mill was panned in small wooden *bateas* for its gold-content.

From the time of the conquest of the Incas by the Spaniards, gold-mining in the Canutillo district seems practically to have ceased until about the year 1700, at which time the industry was revived under Spanish control; and although the old method of mining by incline-shafts, or *chiflones*, continued, there were considerable improvements; iron tools replaced those of wood or stone, and although all the work was done by gads, no traces of drill-holes or blasting is visible. Timbering on a very limited scale was carried on by using the small stunted trees which grew close at hand, and occasionally brush-wood was placed across the stulls so formed, and the waste rock piled on top to save carrying it to the surface. The stoping in nearly all cases was underhand, the original intention being that the timber should support the walls. Forced labor by the natives cost little, and by this means mining was carried on to an average depth of 50 m. wherever fairly soft oxidized ore existed.

For final treatment the ore was taken to Freirina by pack-animals, and crushed in Chilean mills with stone runners and beds, operated by water-power from the Huasco river, with a capacity of about 1 ton per 24 hours; mercury, however, does not appear to have been used. Pyritic ore was unworkable, and on that account, in the few cases where small quantities showed up in mining, the work was evidently stopped. The presence

of water in the mines was an insurmountable difficulty, the only appliance used to remove it being goat-skins, which were filled and carried on men's backs up the *chiflones* to the surface.

Mining work seems to have ceased about the year 1815 during the war of Chilean Independence, probably on account of increase in the cost of labor, supplemented by the discoveries of silver in such abundance at Agua Amarga, in the Huasco valley, as to overshadow all other mining industries near at hand. With the exception of a few isolated attempts at work on a very small scale, the mines were completely abandoned until 1896, in which year the Anglo-Chilean Exploration Co. began its operations.

Geologically, this field is situated at the extreme southern end of an eruption of diorite, about 30 miles long from N. to S., and extending from an extreme distance of 47 miles inland to the shore, and thence westward under the Pacific Ocean. This eruption is similar to many others along and near the coast.

In general, the country-rock is diorite or aphanite, according to the conditions under which it has cooled and crystallized. Around Canutillo the form is coarse pyroxene- (augite) diorite; the plagioclase is labradorite with some anorthite. Diallage and hornblende are generally visible as products of the augite, which, near the lodes, has been entirely converted into the latter mineral. Where the form is aphanite, epidote is often encountered. Quartz is present in small quantities for a considerable distance around the district mentioned; but it appears to be secondary and to have a common origin with the lodes. The average silica-content is about 55 per cent.

About 13 miles up the Huasco valley, and again 32 miles further inland, syenite shows through the diorite, and it is probable that this underlies the diorite to a much greater extent than is visible.

The eruption of the rocks described was partly or wholly submarine; and the more recent gradual upheaval has brought to view entire beds of shells, which show from the shore-line to several hundred feet above sea-level. Close to the contact of the diorite and the hornblende-schist through which it erupted, these shells show in varying degree the effects of the heat to which they have been subjected, according as they are found actually in contact with the diorite, where they are fused into

solid masses with the sand and exist only as casts, or, at some little distance away, in the schist, and so protected from the high temperature. The most numerous of these shells are: GASTROPODA.—*Cerithium plicatum*, *nassa* and *ficula* (comparatively rare); *turbo*, *galerus*.

LAMELLIBRANCHIATA.—*Mactra*.

CALCEROUS SPONGE.—*Rhaphidonema* (abundant).

The order of abundance in which these fossils exist, is—*C. plicatum*, *mactra*, *turbo* and *galerus*.

It appears that the eruption was not earlier than the Upper Cretaceous, and possibly even still later.[1] Since then the denudation has been very rapid and extensive, the surface of the country, sloping at an average grade of 2.5 per cent. towards the Cordilleras, has been cut by numerous gulches which are sometimes almost cañons, being in many places 500 ft. below the surrounding hills. The rainfall, although of late generally limited to about 6 in. per year, varies considerably, and when continuous for more than a few hours the bottoms of the gulches, ordinarily containing only a few water-holes, become the beds of raging torrents in which rocks of a ton weight are carried with ease. At long intervals rains lasting several days almost change the entire face of the country. The hills being practically devoid of vegetation offer a bare surface to the weather, and the denudation is helped by earthquakes which though generally slight are of frequent occurrence. The diorite is decomposed in place generally to a depth of from 8 to 10 m.—a limit which varies greatly with the texture of the rock.

The lodes have a general N.–S. direction, and dip to the West at an average of 75 degrees. As a rule they do not conform to the strike of the country-rock, but cut across it at an acute angle. The lodes exist only in the diorite, ceasing to be well defined as the contact is approached, and in no case do they pass the contact line of the schists.

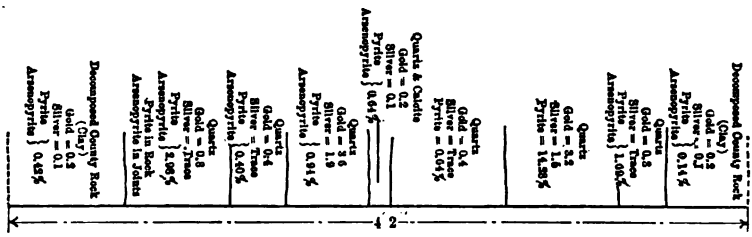
The general structure of the lodes is shown in Fig. 1, which gives a fully developed section. Generally some of the sections are mere narrow seams, but a careful examination shows the total number to be the same in all cases, the difference being

[1] At Guayacan (about 127 miles south of Huasco) deposits similar in fossil-content to the above-named carry in addition bones of some type of whale, but actual evidence of heating by the diorite is not so clear.

the extent to which they are developed, which seems to point to the formation having taken place at four distinct periods, the first three depositing principally quartz, and the last the bulk of the calcite. Explorations below the oxidized zone have not yet gone far enough to prove conclusively which of these deposits carried the bulk of the gold-content, but apparently the last deposit was the poorest. In addition to quartz and calcite, the lodes carry iron, arsenic, sulphur, copper (in very small quantities), gold, silver, and very constant traces of manganese; but no other substances, as far as I am aware.

The lodes vary from a few centimeters to 3 m. in width, with an average of about 0.5 m., and stringers fall into them at frequent intervals, generally joining the lodes horizontally (by

FIG. 1.



NOTE.—The partings have been caused principally by the difference in the crystallization of quartz. They contain no foreign matter except a slight excess of pyrites. Gold and silver in parts per hundred thousand.

SECTION OF A FULLY DEVELOPED LODE AT CANUTILLO, CHILE.

difference of dip instead of strike). These stringers usually tend to augment the gold-content of the lode at their contact, but following them upwards away from the lode has always shown that they pinch to mere narrow seams and decrease rapidly in assay-value as the distance from the lode increases. In all cases the lodes are very well defined; below the oxidized region the gouge-streak on either side varies from 0.125 to 3 in. in thickness and consists of decomposed country-rock in the form of a stiff, dark, slate-colored clay, in which are particles of quartz that have resisted decomposition, and numerous small crystals of arsenopyrite. In general the gold-content is considerably less than that of the adjoining quartz, and only where the lode has given exceptional assay-value is it intentionally included in the ore that is milled.

The walls show signs of alteration by the sericitization of the feldspar, and the replacement of the hornblende by arsenopyrite for a distance from the lodes, varying in proportion to their hardness, but in general limited to about 1 meter. In the case of included country-rock between the lode and a feeder, this alteration is intensified, and often shows the replacement of considerable amounts of the original rock by quartz, in which cases the assay-value may approach that of the surrounding vein-matter, but in no case has the altered country-rock from the walls been found to contain a gold-content of practical value.

In most cases where the lodes are narrow and pass through very hard rock they are poor in gold, and the reverse applies with the exception of extreme widths, which ordinarily carry lower values. Faults have played an all-important part in the original gold-deposition, in fact as far as the present exploration below the region affected by secondary enrichment has shown, it is exclusively in proximity to some fault-plane that workable ore exists, and only so on one side of the fault, when it is generally found with, or close to, small quantities of chalcopyrite.

With the denudation of the country, a secondary enrichment has taken place under the most favorable circumstances possible, which applies equally, or partly, to numerous districts throughout the Republic and apparently accounts for the old idea that "gold does not exist in depth in Chile," as commented on by Darwin in 1846,[2] although at that time the explanation was not so clear.

Naturally in broken country of this character, both the extent of the denudation and the height above permanent water-level depend entirely upon the position of a given point, the latter varying on several lodes from 200 m. on the hill-tops, to a minus quantity where the same lodes run through the bottoms of the deeper gulches. Taking one of the most worked lodes as an average example, with its outcrops at 110 m. above water-level, the denudation by reference to the neighboring hills of exceedingly hard diorite, not likely to have suffered more than a slight weathering, has been approximately 208 meters. At the outcrop the country-rock has been entirely decomposed, and of the lodes only small patches of the more solid

[2] Quoted in *Gold, Its Occurrence and Extraction*, A. G. Lock, p. 232.

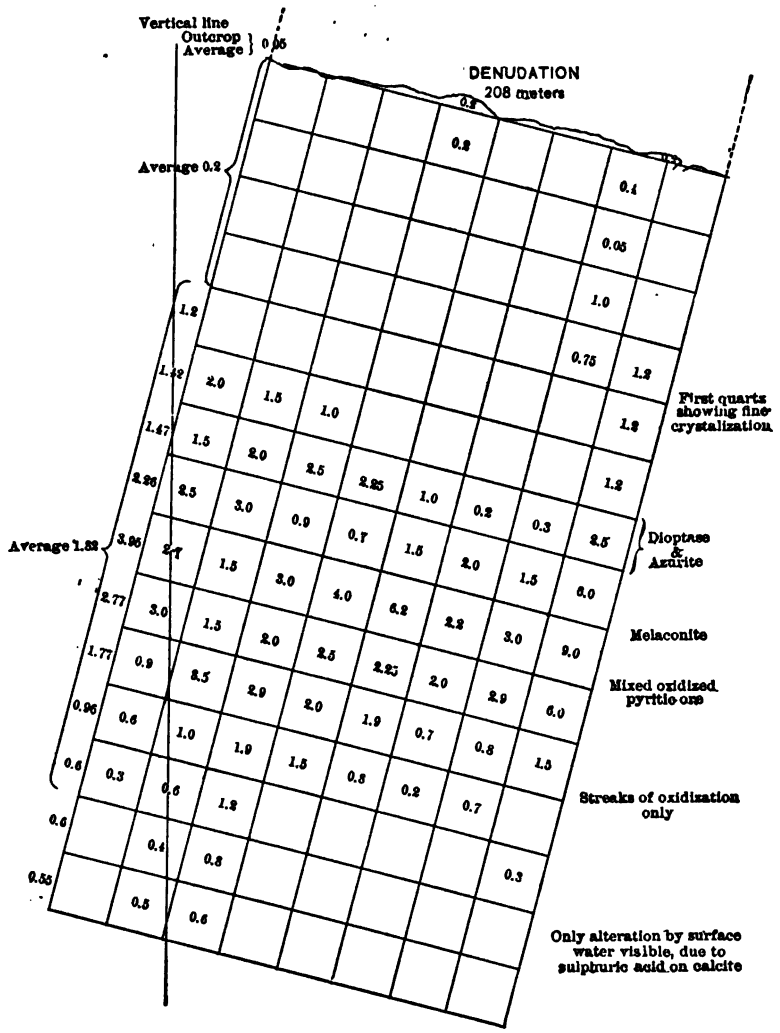
quartz remains, the lode being often only traceable by the stains and decomposed country-rock remaining. Gold is not found in workable quantities at less than 200 m. from the surface, and even then it is present only in small quantities. To about 40 m. the lode consists of a clayey mass containing large quantities of sericite stained with hydrated iron oxide, and small quantities of compact hard quartz; below this is met honeycomb-quartz from which the iron has been so completely removed as to show slight stains only, and in cases leaving perfect pseudomorphs showing the striation of the once-enclosed pyrite, from which part of the gold-content still remains in the cavities, in the form of agglomerated flakes. Farther below is quartz containing decomposed pyrite and arsenopyrite, which in some cases still retain their crystalline structure, although now devoid of both sulphur and arsenic.

At this level, copper first makes its appearance as a stain; the gold, here, is coarser than in the overlying strata, and the ore is of much higher assay-value. Continuing downward, the copper where found, although only in minute quantities, increases in quantity and is in the form of silicate (dioptase) and occasional patches of carbonate (azurite), which latter carries a much larger proportion of silver than the average ore. At about 90 m. the copper met with is oxide (melaconite), and on descending farther, traces of pyrites and arsenopyrite make their appearance, beginning in the less porous portion of the lodes, the oxidized ore decreasing until at 120 m. the only change due to surface water is the decomposition of the calcite by the sulphuric acid formed from the decomposing sulphides (pyrites) above, traces of this action continuing to the greatest depth yet reached. The accompanying sketch, Fig. 2, taken at right angles to the surface, has been selected to show a long stretch of lode uninterrupted by cross-lodes or faults. On it the gold-values given have been averaged from hundreds of assays, and show the regular increase in the value of the ore from the surface down to a point where the bulk of the deposition from descending solutions might reasonably be expected to take place, and from this point the steady decrease until the original assay-value of the ore is reached, which as far as experience has gone remains general and constant.

In my opinion, under the extremely favorable circumstances

which have existed for secondary enrichment by rapid and extensive denudation with just sufficient water to produce a slow

FIG. 2.



NOTE.—Each section corresponds to an area of 10 m. square, and the included numbers represent the gold-content in parts per hundred thousand.

SECTION OF THE PERSEVERANCE LODE AT CANUTILLO, CHILE.

seepage through the ground, the gold which existed throughout all the lode to the extent of 0.55 parts in 100,000, has been dissolved from the denuded portion of the lode almost com-

pletely, probably by the sulphuric acid liberated from the decomposing pyrites acting on the chlorides present principally as common salt, further aided by the presence of manganese; and as the walls have been practically impervious to water, the solutions have percolated almost entirely down the lode itself, precipitating the gold at lower levels where suitable substances existed; this leaching having proceeded faster than, or at some distance in advance of, the denudation of the surface of the country. In this manner can be explained the presence of small quantities of gold in the outcrops, and the fact that no alluvial gold has been found except as traces in the gulches, although several shafts have been sunk to bedrock in likely places. Moreover, starting on the assumption that the original lode extended from the present unaltered region to 208 m. above the outcrop (Fig. 2) and calculating the concentration that has taken place, the results given in Table I. are obtained.

TABLE I.—*Original and Present Distribution of the Gold in the Canutillo District.*

	Height.	Length.	Thickness.	Volume.	Specific Gravity.	Weight.	Gold-Content.	Total Gold-Content.
	Meters.	Meters.	Meters.	Cu. M.		Metric Tons.	Pts. per 100.000.	Kilograms.
Original lode.....	338	80	0.416	11,248	2.6	29,244	0.55	160,842
Denuded from surface after leaching.	208	80	0.416	6,922	2.6	17,997	0.05	(a) 8,998
Leached portion of lode.....	40	80	0.416	1,331	2.0	2,662	0.20	5,324
Enriched portion of lode.....	90	80	0.416	2,995	2.6	7,787	1.82	141,643
Difference, unaccounted for.....	.....	.....	.....	.....	.....	.....	.....	4,877
Total.....	.....	.....	.....	.....	.....	.....	.....	160,842

(a) Lost in denuded portion.

It seems also probable that, following the leaching of the gold, the quartz itself has been decomposed, except where present in solid masses, for the reason that only in rare cases do the outcrops show more than a small fraction of the amount of quartz existing at 40 m. or deeper. The absence of all but compact quartz, both from near the outcrops and as float rock,

FIG. 3.

a, Wire-Rope Tram-Way.

HEAD-GEAR AT THE PERSEVERANCE MINE, SHOWING PART OF ORE-FLOOR, AND WIRE-ROPE TRAM-WAY TO MILL.

FIG. 4.

a, Wire-Rope Tram-Way from Perseverance Mine.
GENERAL VIEW OF THE MILL, LOOKING SOUTHWARD.

does not seem accounted for otherwise, while its decomposition seems possible if the high alkaline character of the surface-water be taken into consideration.

The mining of the ore presents no features of interest; the ground is good, and shafts, or levels, on the lodes require nothing more than light timbering, due to the absence of swelling or strain. The quantity of water in the lower levels is little more than enough to keep the ground saturated, and in the upper levels it is somewhat less. The ore is overhand-stoped by hand-labor, and generally the wall-rock brought down is sufficient to fill in the waste space as the stopes rise.

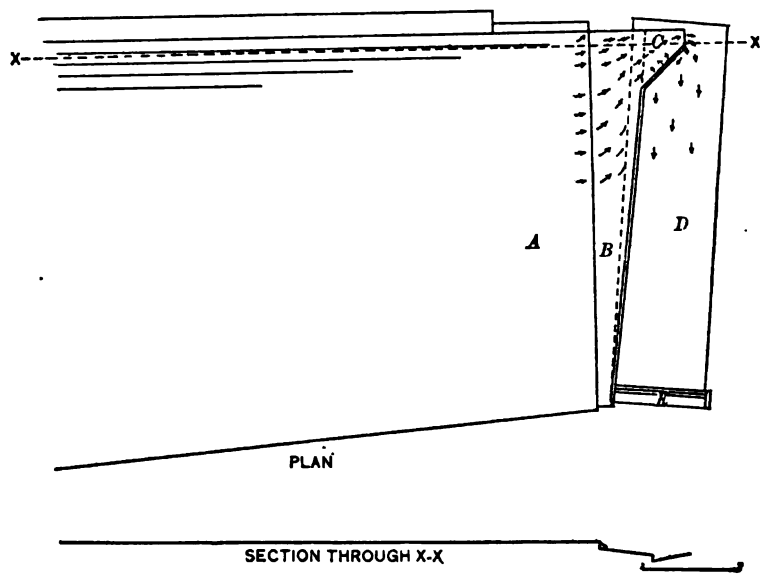
The half-tone illustrations, Figs. 3 and 4, show the head-gear at the Perseverance mine and a general view looking southward from the mill, to which the ore is transported by wire-rope tram-way from the mine.

The ore is sorted at the ore-floors in the mines, and then sent to the mill. From the ore-floor of the mill, all of the rock is passed through a 10- by 8-in. Blake crusher set to crush to a 2-in. ring, and falls into a bin placed below which serves for the common feed of three Tremain steam-stamps. These machines are described in our *Transactions*;[3] but like most patent grinders and crushers that have been designed as improvements on the gravity-stamp, they fall a long way behind it in the crushing and amalgamation of gold-ores. The steam-cylinder being directly over the mortar-boxes, allows leakage of oil through the glands which finds its way into the mortar unless great care is taken, and even then small quantities get through. In common with most machines controlled by valves actuated by direct steam, they are subject to erratic stoppages, sometimes from causes so slight that only after careful search can the trouble be placed,—a disadvantage which is aggravated in the case of the stamp, by the vibration to which it is continuously subjected. Most important of all, however, is their excessively large consumption of steam, the total for the mill amounting to 1 ton of good-quality coal to 7 tons of ore crushed, inclusive of the quantities required to operate the rock-breaker, to pump the battery-water and to run the ore-concentrator. The steam is condensed and returned hot to a Cornish boiler which supplies the mill-plant.

[3] *Trans.*, xxvi., 545.

The mortar-boxes are provided with back-plates and with lip-plates fixed outside immediately under the screens. Mercury is added in quantities varying with the richness of the ore, the average being 60 g. per 24 hours. Steel-wire battery-screen of 30 mesh is used and the discharge maintained at 4 in. above the dies; under these conditions each battery of two heads crushes 10 tons per 24 hours.

FIG. 5.



Notes
A Wilfley Table. Concentrates end (Plan)
B Amalgamated plate, slopes outward and to discharge end
C Shovel shaped plate, movement of which unites particles of mercury
D Independent plate below discharge of C
E Mercury Trap

ARRANGEMENT OF TABLE, PLATES AND MERCURY-TRAP AT THE MILL, CANUTILLO, CHILE.

From the lip-plates the pulp passes through a mercury-trap 4 ft. wide by 6 in. across, containing a bath of mercury 1 inch deep. Across the bath is a center-board which causes the pulps to dip once below the surface, thus serving principally as a distributor from the front- and side-discharges of the mortars to the plates, which are 8 ft. long by 4 ft. wide, set with 1.25 in. fall to the foot. Beyond the plates are traps similar to the first one, followed by a Wilfley table, which has at the head- or concentrates-end the arrangement of plates and traps shown in

Fig. 5. After dropping from the end of the table the concentrates pass over these additional plates and traps and effect a saving of 7.08 per cent. of the total mercury loss (which amounts to 0.0325 kg. per ton of ore milled), and 0.68 per cent. of the total gold-content of the ore in the form of amalgam.

The concentrates, amounting to 1.4 per cent. of the total quantity of ore milled, are sold for smelting. The tailings from the concentrator pass through a pointed-box settler and discharge into the gulch for possible future treatment, while the clear water is returned to the battery. The average distribution of amalgam is as follows:—Back-plate, 8; around dies, 15; front and side lip-plates, 42.3; traps, 20; apron-plates, 12; concentrator plates and trap, 2.7; total, 100 per cent.

These figures show that only a total of 23 per cent. of the amalgam is caught inside the batteries; this small proportion is due to the rapid blows of the stamps (160 per head per minute), and the consequent violent agitation inside the mortars, which to a large extent accounts for the high consumption of mercury, due to flouring. The quantity of gold yielded by the amalgam varies greatly, being more from the lip-plates and battery than from the traps and aprons; furthermore, the total yield varies according to the physical condition of the gold, whether comparatively coarse, as is the case with rich ores; or fine, with poor ores. The general average yield in bar-gold is 40 per cent. of the amalgam retorted. The composition of the bullion approximates 830 fine in gold, 115 fine in silver, the balance being principally copper.

Practically all the gold is free and amalgamates very readily; even though pyrite be present in considerable quantities fine grinding and careful panning or spooning of the ore will show gold if present to the extent of 0.3 oz. per ton, or more. The gold, however, is not coarse except in rich samples, as it is rare to find metallic grains that will not pass a 100-mesh sieve from ores that assay up to 1 oz. per ton. Silver exists in the ore in the proportion of about 1 of silver to 3 of gold, but only a small part of this is alloyed with the latter metal, which, separated alone by washing, assays over 95 per cent. of gold.

Immediately below the discharge of the Blake crusher is set a sampling machine, which to all practical intents and purposes is a small elevator-belt, driven from the countershaft of

the crusher, placed nearly horizontal and carrying two small elevator-buckets so arranged that they pass under the discharge opening and then dump their contents into a box placed behind the crusher, the pulley being far enough back for the falling ore to pass just clear of the belt, but be caught by the buckets, which take a total of 0.3 per cent. of the total quantity of ore crushed. The samples are collected, the product from each mine being kept separate. At the end of the month these samples are hand-crushed and quartered down to assay-packet size and the process repeated, giving independent double samples of each lot taken by the machine. Small rich lots are weighed, sampled by repeated quarterings, and assayed separately, this precaution being taken in order to prevent the coarse gold from interfering with the assays when added to the bulk.

The tailings, after leaving the concentrator and before entering the settler, pass through a tailings-sampler of the Lamb type working continuously, the collection of each two days' work by the machine being separately assayed.

The concentrates and the bullion are weighed, sampled and assayed, and by this means a check is kept on the working of the mill which has up to date been entirely successful, the gold-content of the ore milled balancing that of bullion, concentrates and tails, to within about 0.5 per cent. The difference generally showing as a persistent small surplus of gold, somewhat difficult to account for.

The amalgamation results taken on about 8,000 tons of ore milled are as follows:

Metric Tons.	Fine Total Gold-Contents.	Fine Gold in Bars.	Extraction.
	Kg.	Kg.	Per Cent.
Ore milled.....8,257.016	180.492	151.550	= 83.96
Concentrates produced, 116.309	8.530	.....	= 4.72
Total extraction.....	.....	.....	88.68

Notes on the Flow of Gas from Orifices.

BY W. R. CRANE, UNIVERSITY OF KANSAS, LAWRENCE, KANSAS.

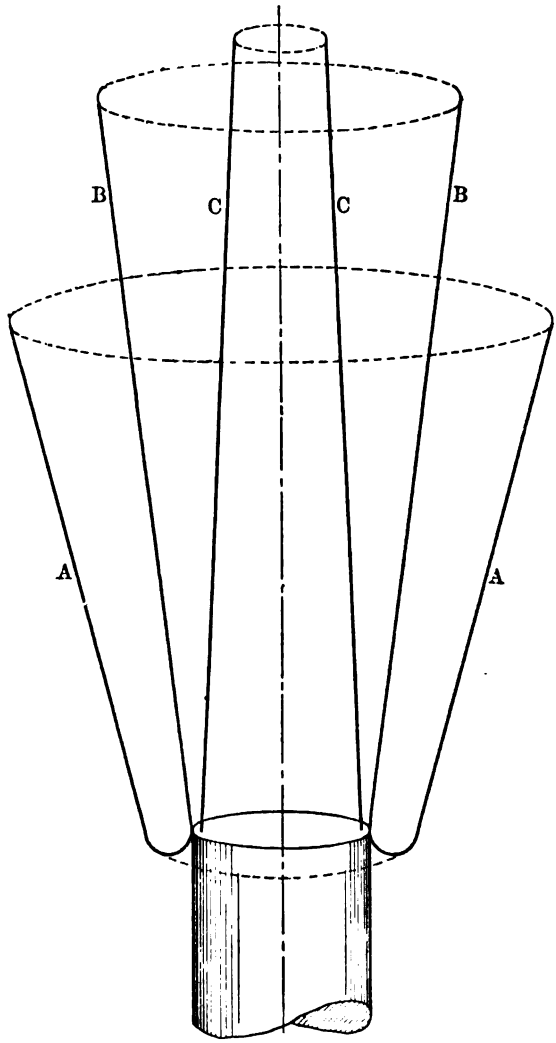
(Lake Superior Meeting, September, 1904.)

WHILE professionally engaged in the oil- and gas-fields of Kansas, in measuring the pressure and flow of gas-wells, and studying the phenomena attendant upon the production of both oil and gas, I noticed in the flow of gas a peculiar and interesting feature.

That there are actual lines of flow in liquids (non-compressible fluids) discharged through an orifice, or a short section of pipe, is shown by the external shape of the discharge—the *vena contracta.* Such lines of flow are not observable in the discharge of gas from an orifice. It can hardly be questioned that they exist. That they are not apparent is probably due to the very small weight of the gaseous particles, which diminishes the momentum of their movement in a given direction, acquired before they issue from the orifice, and also to the expansion of the gases, which would exert a powerful influence in overcoming any such tendency in their flow.

Gas flowing from an orifice, supplied by a large reservoir or blowing-engine, exhibits almost the reverse of what is noticed with liquids. Namely, the gas assumes the form of an inverted truncated cone, of large angle, with its apex-end at the orifice. (See B B, Fig. 1.) This external shape assumed by the flow of gas shows, however, marked differences under varying conditions of volume and pressure. For instance, a large volume of gas under high pressure, flowing through a 2- or 3-in. pipe, as in the case of a natural gas-well, assumes a more definite conical form, the angle of which increases as the pressure falls. Moreover, outside of the well-defined inverted cone is a thin conical sheath or zone of gas, A A, the angle of which is considerably greater than that of the main cone, and increases with

FIG. 1.



A A, Misty retarded gas, most visible when much moisture is present. B B, Densely smoky rarified gas, always visible. C C, Nearly transparent unrarified gas, visible only when little or no moisture is present.

ZONES OF GAS-FLOW.

the pressure; while within the main inverted cone, B B, there is still another cone, C C, standing, apex-end upwards, upon a base of slightly smaller diameter than the orifice. The angle

of this inner cone is very small compared with the others, and, like that of the main cone, varies inversely as the pressure.

The outermost cone (A A, Fig. 1) can be distinguished from the main or intermediate cone, B B, by the much smaller density of its gas. In fact, it is made up largely of the outer portion of the escaping stream of gas, retarded in passing over the sharp inner edge of the orifice, and is acted upon by the propellent and expansive forces of the adjacent stream, which force it outward and downward. It thus assumes the form shown in the diagram.

The contrast between, A A, and, B B, is very marked, especially when considerable water is present in the gas, and when large volumes of gas under high pressures are discharged. Even more marked is the contrast between the zones, C C, and B B, but the cause of this is quite different.

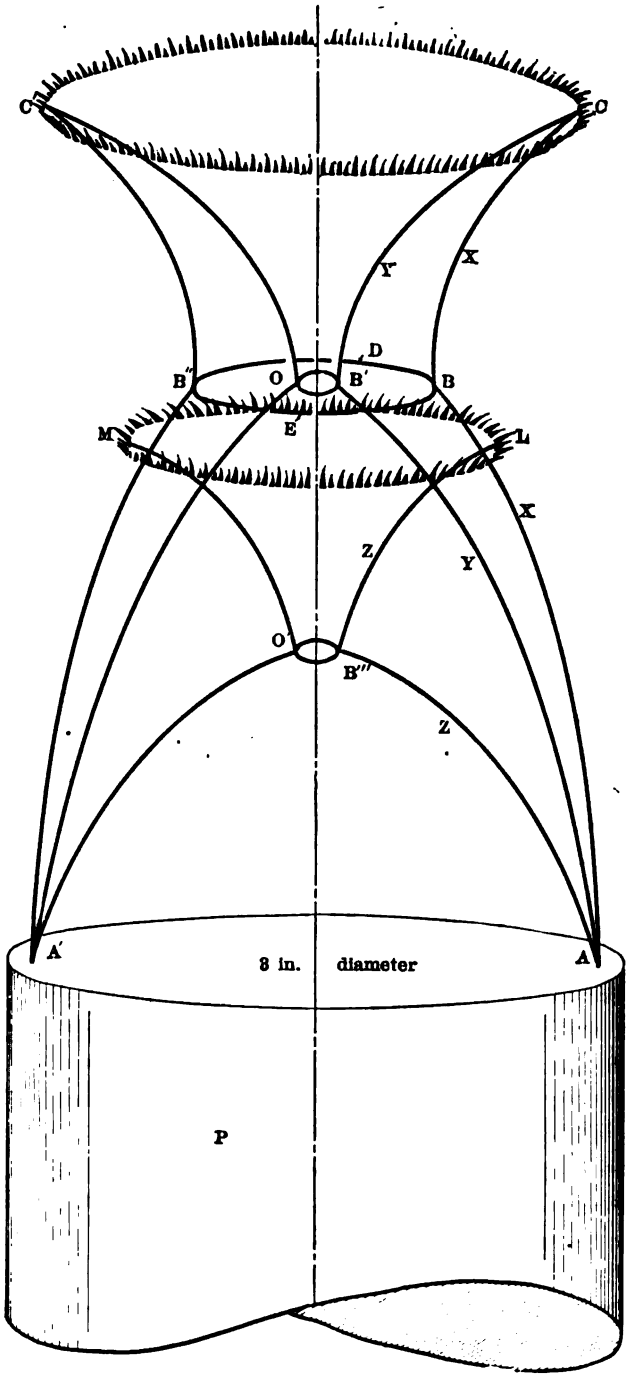
Gas-wells are, as a rule, quite wet, often raising considerable water *en masse* during flow. But sometimes the amount of contained water is very small and invisible, so that the well is considered as practically dry. In the former case, the whole issue of gas may be so charged with water as to assume a smoky appearance; in the latter case, the gas is transparent until its temperature has been sufficiently lowered to cause the segregation of aqueous vapor. Now, comparatively dry gas, on escaping from the orifice, is at once subject to expansion, which begins at the mouth of the orifice and continues through the upward flow, with results which are mainly apparent laterally, in the formation of the rarefied exterior envelope, and the relative compression of the innermost cone. By reason of the latter action, the central cone of denser gas converges upward. The main cone, between the inverted exterior and upright interior cones, consists of partially expanded gas, which will hold its water invisible so long as it remains above a certain limit of pressure, approximating the pressure of the reservoir; but a sufficient fall in pressure, and corresponding fall in temperature, cause a condensation of the contained water. Hence, on looking at a flowing well against a dark sky, a central upright cone, C C, of clear gas is seen, surrounding which is a conical inverted sheath, B B, of gas, rendered misty by the condensed vapor, and encircled in turn by a much less clearly defined conical zone, A A, often nearly invisible, of retarded gas. These

several zones of gas are often noticed within wide limits of pressure and volume.

Another phenomenon is noticed at high, which cannot be seen at low pressures, namely, a peculiarly-shaped figure or series of figures (Fig. 2), made apparent by the condensation of the gas, and due, probably, to lines of flow. These figures are found in the inner upright cone (C C, Fig. 1), and, beginning at the mouth of the orifice, recur upwards with gradually diminishing intensity, until they are no longer visible. They are remarkably uniform in shape, and vary with pressure and volume only. I have observed and studied them in wells with capacities ranging from a few hundred to 10,000,000 cu. ft. per 24 hr.; but in no case have I seen them in wells having a flowage-pressure of less than 8 or 10 lb. per sq. in. To be visible, therefore, in wells of small capacity, they must be looked for only when the well is first opened to the atmosphere, that is, while the gas is expelled under an accumulated pressure, which may be very high at that period, even for a well subsequently proving to be of small productiveness.

These figures, as observed under conditions most favorable for maintenance of form, are shown in Fig. 2. The body of the figure, A B B″ A′, the first above the orifice, which is the only one completely visible, is not unlike an ellipsoid of revolution, which has been intersected by two parallel planes, at right angles to the major axis, one constituting the base, and the other cutting off a portion of the upper end. Attached to the truncated portion is an inverted conoidal projection, C B B″ C′, a continuation of the part below, the upper diameter of which is about equal to that of the main body of the figure as measured at the mouth of the orifice. The extreme upper part of the figure gradually fades away until it becomes invisible, while that portion of the conoid below reaches a point where it is most clearly defined, *i.e.*, at the junction of the two parts B B″. At this place a well-defined band (B E B″ D) is visible, which expands and contracts with the slightest change in pressure of the flowing gas. It resembles to a marked degree a small, transparent, elastic band, suspended in mid-air, and caused to expand and contract by some invisible force.

Viewed under favorable conditions, these figures resemble remarkably the form of an old-fashioned porcelain egg-cup, ex-



C B B″ C′, A B B″ A′, and united at the plane of constriction B E B″ D, form the figure (marked X) for highest pressure and volume given. As these diminish, the figure passes through the forms A B′ C C′ O A′ (marked Y) and A B‴ L M O′ A′ (marked Z), and finally disappears into P.

FIG. 2.—FLOWAGE-FIGURES AT DIFFERENT PRESSURES AND VOLUMES.

cept that the sides, C C′, of the upper part are concave and not convex (as in the egg-cup). Such a cup, made of glass, and placed, small end up, in a glass cylinder full of water, would closely resemble in appearance the figures here described—the differing refractive power of the water and the glass producing an effect like that which distinguishes the figures under favorable conditions.

When observed in a well of small capacity (say, with a 2-in. orifice) the phantom figure appears with the first rush of gas and the band is distended to the diameter of 0.5 or 0.75 in. (depending largely on the size of pipe and the pressure); then, as the pressure falls by reason of the exhaustion of the accumulated head, it begins to contract until a diameter of 0.125 in. is reached, when the figure begins to shorten and finally disappears wholly within the pipe, the figure changing its shape to conform with the position of the band. Fig. 2 shows three positions of the compound form described, namely, X, the one already discussed, which is the largest, and appears at highest pressure and volume; Y, a position assumed at considerably lower pressure and volume; and Z, the position of the form just before, under a great reduction of pressure and volume, it disappears, withdrawing itself, as it were, into the pipe P.

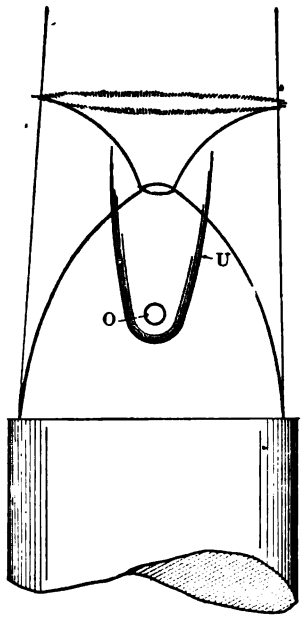
The figures cannot always be seen even in the same well; since the pressure may fall to such an extent that they will not be prominent enough to be visible; or they may be hidden from view by moisture brought up in the form of spray or mist by the escaping gas. In the former case, it may be possible to bring them into prominence again by constricting the size of the orifice, as, for instance, by changing a 3- for a 2-, or a 2- for a 1-in. pipe, which can be connected by a reducing-plug to the top of the control-valve.

Seldom are more than two well-defined figures observable, the second being superimposed above the first; but occasionally a third figure can be seen, superimposed in turn above the second, etc. That still others exist is indicated by the mottled appearance of the inner upright cone.

When an obstruction is introduced into the flow of gas from a well, the gas is compressed on the sides of the obstruction in contact with the current, thus forming a U-shaped figure, rendered visible by the unequal refraction of light in passing

through gas of different densities (a cause doubtless operating also in distinguishing the zones and figures in Figs. 1 and 2). This phenomenon is indicated in Fig. 3. If the obstacle is a rod placed parallel with the flow, a cup-shaped figure is produced about the end of the rod. This was first noticed in taking measurements with the Pitot tube, as described below. An attempt was afterwards made, by inserting a Pitot tube, to break up the figures of Fig. 2; but it only produced a slight distortion.

FIG. 3.



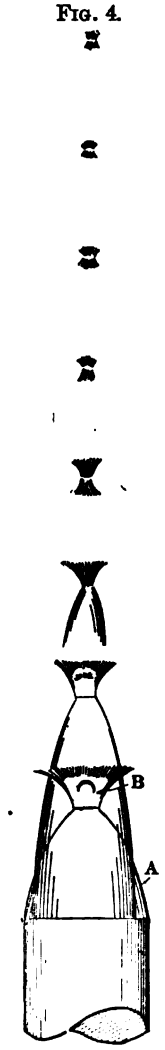
O, Obstacle. U, U-shaped figure.

ADDITIONAL, U-SHAPED FIGURE, PRODUCED BY INSERTING AN OBSTACLE INTO THE GAS-CURRENT.

In view of the many difficulties encountered in studying the phenomena of Figs. 1 and 2 in gas-wells, an attempt was made to produce them by artificial means. After many trials with different apparatus, such as air-pumps, air-compressors, etc., the figures were finally produced in the discharge of a quadruple (4-stage) air-compressor (liquid air-machine), at the end of the small copper tube, which connects with the liquifier:—connection being broken at this point. The internal diameter of the

tube is 0.125 in., and although the figures were small they were quite distinct, especially the first one. The compressor was run at a pressure of 350 lb. per sq. in. Fig. 4 shows the result.

FIG. 4.



Enlarged about five times. The first two figures were quite distinct, and the next two fairly so: the rest being indicated as shown.

FIGURES PRODUCED IN THE DISCHARGE FROM A FOUR-STAGE AIR-COMPRESSOR.

A close study of the figures thus produced served as a check upon earlier observations, verifying several points that had not been definitely and satisfactorily determined, and bringing out other points not previously observed. Measurements were taken with a Pitot tube across the mouth of the orifice, and also at fixed distances from the orifice. The smallness of the orifice necessitated the making of a special Pitot tube, which, by reason of its size, was made of glass, to give it rigidity, and was drawn to a very small opening. This tube was clamped in various positions and opposed to the flow of the air. A mercury-gauge was employed in measuring the pressures obtained in the tests.

An attempt was made to measure by the Pitot tube the pressure at various points in the figures, especially in the first and most clearly defined one, but with little success. The figures were so small, and the Pitot tube so large in comparison, that the former were distorted, and the resulting measurements were vitiated to such an extent that the work was temporarily abandoned, though not until an attempt had been made to enlarge the figures by enlarging the orifice. Tubes of different sizes were attached; but the figures failed to make their appearance, except in a tube of 0.125-in. diameter—the size of the original orifice.

A series of Pitot-tube measurements, made with the different-sized tubes, are given in the table on the following page.

No.	Diameter of Orifice in Inches.	Flowage-Pressure in Inches of Mercury.							
		Position of Pitot Tube at Mouth of Orifice.				Distance of Pitot Tube from Orifice (in Center of Flow).			
		At Center.	One-third of Radius from Center.	Two-thirds of Radius from Center.	At Side of Orifice.	1 In.	2 In.	3 In.	4 In.
1.....	$\frac{1}{8}$	32.22	32.08	32.00*	.....	28.02	25.50	24.08	4.17
2.....	$\frac{11}{64}$	29.50	28.12	25.00*	.....	28.08	17.00	6.00	1.06
3.....	$\frac{15}{64}$	28.02	26.18	26.12*	.....	14.14	6.02	2.14	1.00
4.....	$\frac{21}{64}$	6.18	6.12	6.08	6.02	6.06	3.00	2.00	0.14
5.....	$\frac{31}{64}$	4.02	4.00	4.00	3.95	6.00	4.50	4.50	3.00

* These figures really belong in the next column to the right, owing to the small orifice and relatively large Pitot tube. While these tests were being taken the compressor was working at a pressure of 350 lb. per sq. in. The mouth of the Pitot tube had an internal diameter of $\frac{1}{64}$th of an inch.

The variation in temperature with distance was noted as follows:

Distance (inches), . . .	1	2	3	4	5
Temperature (degrees C.), . .	7	10.5	14	15.5	17

The lined appearance of the first figure, as shown at A, Fig. 4, seems to indicate that all of the figures have their origin at the mouth of the orifice, each succeeding figure having a length equal to that of the one immediately preceding, plus a constant quantity, which is (as closely as could well be determined) the distance from the constricting ring of the first figure to the mouth of the orifice.

The well-defined upright conical zone, noticed in gas-wells, was not present in the latter tests; but the gradual reduction in the size of successive figures clearly indicates its presence.

The peculiar detached figure or portion of figure, B (Fig. 4), was not noticed in the gas-wells, but was brought out by the later experiments. It was so extremely small that no attempt was made to investigate it.

As a mere outline of the possible causes of the phenomena above described, the following suggestions are offered as they occurred to me:

1. That the figures are nodes produced by sound-waves. This was proved untenable by attempts made at gas-wells to shorten the nodes by breaking the column, and in the later experiments, when no sound accompanied the production of the figures.

2. That they are flowage-figures, attendant upon the flow of gases under all conditions, but intensified and rendered visible only under conditions of high pressure and volume. The effect of both pressure and volume, and the instantaneous response in change of form to the slightest variation of either, show without doubt that the figures owe their existence to the flow of the gas. The extremely variable state of the gas in the reservoir and the conduit leading to the orifice, together with the manifold forces acting upon it when it merges into the atmosphere, render the tracing of the path of a particle of gas not only very difficult, but, at best, uncertain.

3. The hypothesis that the curve, A B, Fig. 2, is the path or trajectory of a particle of gas, has been considered, but, by reason of the lack of sufficient reliable data, it cannot be substantiated.

I offer, therefore, no positive explanation of the phenomena, but simply a description of them, which may lead to further investigation and discussion. The subject seems to possess, at present, no economic importance, though it is extremely interesting physically. But who can say how soon a purely scientific observation may acquire great commercial value?

The Fire-Clays of Missouri.

BY H. A. WHEELER, ST. LOUIS, MO.

(Lake Superior Meeting, September, 1904.)*

The Clay Industry of the United States.

It may surprise some of our members to learn, that, among the industries based on the mineral resources of the United States, clay now ranks third, being exceeded in value of product only by pig-iron and coal. The present annual value of clay-products exceeds the value of the outputs of gold or silver, and also exceeds the combined values of copper, lead, zinc, aluminum, mercury, antimony and nickel. These facts are shown in Table I. which gives the statistics for the year 1902:

* Presented at the supplementary session of the Institute in St. Louis.

TABLE I.—*Production of Metals, Coal and Clay in the United States During* 1902.[1]

	Value.
Pig-iron,	$372,775,000
Coal (including anthracite, $76,173,586),	367,032,069
Clay products,	122,169,531
Gold,	80,000,000
Copper,	76,568,954
Petroleum,	71,178,910
Silver (coin-value),	71,757,575
Lead,	22,140,000
Zinc,	14,625,596
Aluminum,	2,284,590
Mercury,	1,467,848
Antimony,	634,506
Nickel,	2,701

NOTE.—The value of the clay-products in the United States in 1903 was $130,962,648.

Of the total amount, $122,169,531 assigned for the value of the clay-products of the United States during 1902, about 20 per cent. is classified as pottery, or high-grade goods, and 80 per cent. as brick and tile, or low-grade ware. The latter is again subdivided into building-brick, vitrified brick, fire-brick, sewer-pipe, terra-cotta, tiles and tiling, etc. The total value of the fire-brick produced in 1902 was $12,601,435, or about 10 per cent. of the value of the entire clay-products. Seven States furnished nearly 90 per cent. of the total quantity of fire-brick, which includes also stove-linings, fire-tiling, retorts, tuyeres, and other shapes produced by fire-brick manufacturers.

THE PRODUCTION OF FIRE-BRICK.

The comparative value of the fire-brick industry in 1902 is given in Table II. on the following page.

The State of Pennsylvania ranks pre-eminently first in importance in the refractory-brick industry, and it is always likely to retain this position, as it is very rich in an extensive area of high-grade fire-clay in the central part of the State, and another large area of nearly as high-grade fire-clay in the southwestern portion, both of which occur at the base of the Coal Measures.

[1] *Mineral Resources of the United States for* 1902, United States Geological Survey, Washington, pp. 38 and 707.

TABLE II.—*Production of Fire-Brick During* 1902.

Order of Importance.	Name of State.	Value.
1.	Pennsylvania,	$6,080,213
2.	Ohio,	1,327,982
3.	New Jersey,	819,580
4.	Missouri,	739,385
5.	Colorado,	609,495
6.	Kentucky,	605,448
7.	New York,	402,006
8.	Maryland,	277,290
9.	Alabama,	222,660
10.	Illinois,	199,048
11.	Montana,	113,112
12.	Other States,	1,179,740
	Total,	$11,970,511

These natural advantages, coupled with the best of cheap fuels, an enormous home-market in the iron-, steel- and coke-districts of Pittsburg and the Lehigh valley, and very low freight-rates, give it an unapproachable advantage over all other States. In this connection it is well to note that the consumption of fuel for making, drying and burning fire-brick is a large factor of their cost. Under the stimulus of the recent enormous expansion of the iron- and steel-business in the United States, the value of the fire-brick output in Pennsylvania has grown from $3,000,000 in 1898 to $6,080,213 in 1902.

Ohio, ranking second in the value of clay products, possesses an abundance of fair-grade fire-clays, has excellent cheap fuel, low freight-rates, and a large home-market in the iron- and steel-plants scattered throughout the State from Cleveland to the Ohio river; to a much less favorable degree, however, than Pennsylvania. The plants in southern Ohio adjoin the superior flint fire-clays of Kentucky, and promise to stimulate the production of the numerous fire-brick factories along the Ohio river.

New Jersey ranks third in importance, and although the oldest of the fire-brick producers, it is being slowly but surely displaced by Missouri, Colorado and Kentucky. New Jersey has a variety of clays of Cretaceous age, many of which are refractory. Although the clay-producing district is limited in extent, the beds are usually thick and the quantity still available is very large. The clay-beds are at tidewater, and very low freight-rates are thus assured on clay- and fire-brick shipments

to most of the large territory extending from Boston to Baltimore. Fuel, however, is somewhat high, when compared with Pennsylvania or Ohio, while the home-market, though large and varied, has probably reached if not passed its zenith, and will always be menaced by foreign competition.

Missouri ranks fourth, and is growing rapidly in spite of having its home-market heavily curtailed by the exhaustion of its important iron-ores. Twenty-five years ago there were twenty blast-furnaces, a large Bessemer plant, and several iron-mills in and about St. Louis, but the exhaustion of its once world-renowned beds of iron-ore at Pilot Knob and Iron Mountain has reduced this local iron- and steel-industry to two active blast-furnaces. While this staggering blow discouraged the development of new fire-brick plants, the old plants have been improved and enlarged. By developing the higher grades of ware, they are extending their goods to greater and wider horizons, and building up a broad business that is, at least, more healthy and reliable than when dependent upon a local specialized market. According to the statistics of the United States Geological Survey, the value of the output of fire-brick in Missouri has grown from $268,173 in 1898 to $925,915 in 1903, though local returns indicate a much larger growth than this.

Colorado has recently pushed rapidly ahead to fifth place, under the stimulus of the rapid growth in the Pueblo iron-, steel- and coking-plants, and the concentration of the lead- and copper-smelting industries to the Rocky Mountain districts. Superior fire-clays of Cretaceous age occur along the eastern base of the mountains, and while fuel is inferior and high, the local industry is protected by the heavy freight-rates incident to a 1,000-mile up-grade haul from St. Louis, its nearest competitor.

Kentucky, now sixth in importance in the value of the annual output of clay-products, is increasing rapidly. This State possesses at the base of the Coal Measures a very high-grade fire-clay that covers a large area, and with which is associated cheap, fairly good coal. The home-market, though small, is growing, and goods of high quality can be produced that will bear considerable transportation, especially as the railroads are disposed to be liberal in their freight-rates.

New York, ranking seventh in the order of importance of clay, is merely the extension of the New Jersey clay-field, and therefore labors under the same disadvantages possessed by New Jersey.

Maryland is of more importance historically than from its present or prospective output, although it possesses very superior fire-clay at the base of the Coal Measures in the Mount Savage district. The value of the output in 1902 amounted approximately to $300,000, and the industry is slowly diminishing, although the famous Mount Savage fire-brick has been on the market for more than 60 years. The competition of districts nearer to the iron- and steel-centers has interfered with the growth of the refractory-brick industry in Maryland, which is consequently limited to special classes of products.

The Missouri Fire-Clays.

The Missouri fire-clays are divided into two classes—the flint fire-clays and the plastic fire-clays—each having very different properties and geological occurrence. Both are used by Missouri fire-brick manufacturers, and each has special merits when properly used. The flint fire-clays are exceptionally pure, are devoid of plasticity, and if their excessive fire-shrinkage is removed by a preliminary burning, they make the most refractory or infusible fire-brick that it is possible to make out of clay, at least a brick that cannot be fused or injured by the highest heat attainable in a furnace fired by coal or coke. Although these bricks are ideal for withstanding heat, they are very unsatisfactory in shaft-furnaces or under conditions involving abrasive action. Their tensile-strength or cohesion is so slight that fire-brick made exclusively of flint-clay rapidly wear out from the abrasive action of the descending charge. In the arch of an open-hearth furnace, however, or in the checker-work of a regenerator, where only a "dry" or gaseous heat has to be withstood, they are very durable and satisfactory, notwithstanding the very high temperature attained.

The plastic fire-clays of Missouri are tough and plastic, and are only fairly fire-resistant on account of impurities present. These clays make a strong, durable brick that admirably withstands mechanical wear or abrasion, and are excellent for making gas- and zinc-retorts, linings for shaft-furnaces, lime-kilns,

etc., but they cannot resist extremely high heats without "letting down" or softening, or fusing.

By skilfully combining the flint- and the plastic fire-clays in proper proportions, the experienced fire-brick maker can produce a brick that possesses ample refractoriness combined with a maximum strength or toughness, providing the customer clearly defines the conditions under which the brick is to be used. For most purposes the best fire-brick is composed of more or less pre-burned flint-clay, or "grog," the body or bond consisting of plastic clay. Too frequently the metallurgist considers only the refractoriness, or the purity as shown by a chemical analysis of a fire-brick, and overlooks the mechanical weakness caused by the use of an excessive quantity of the non-plastic flint-clay; as a result, the bricks are rapidly cut out by abrasive action. Although not all of the fire-brick makers are equipped with modern testing-laboratories using Seger-cones or Le Chatelier pyrometers, in most cases the manufacturer is better qualified than the metallurgist to design a fire-brick to meet given conditions; and longer campaigns and fewer re-linings of furnaces would result if this responsibility were transferred to intelligent, up-to-date manufacturers. Some of the Missouri fire-brick makers do not belong to that class, but continue to claim, as did their revered grandfathers, a heat-resistance of from 4,000° to 5,000° F. for their products, although the Le Chatelier pyrometer shows them to be unsafe at 2,600° F.

The St. Louis Fire-Brick Market.—The St. Louis market furnishes ten well-defined grades of fire-brick, ranging in heat-resistance from 2,300° to 3,000° F., and in value from $14 to $35 per 1,000 f.o.b. cars at factory. These fire-bricks are manufactured by the old hand-press or slop process, and by brick-machines of the auger and semi-dry types. With regard to the respective merits of hand-made vs. machine-made brick, the old hand-press unquestionably makes a more refractory and superior brick, but the machine-made brick is more uniform and economical, and is now made with such care as to be quite satisfactory. The machine-made brick is also becoming of much greater importance in these days of labor troubles, by greatly reducing the quantity and skill of the labor required for a large output.

The marked range in refractoriness and price of fire-brick gives no clue whatever of the value to the metallurgist, since the cheapest brick, which is made entirely of plastic clay, would prove far more durable in the hearths of roasting-furnaces, linings for water-jacketted furnaces and lining of lime-kilns, where abrasive action is severe, than those of higher price; but the heat-resistant quality of the cheaper brick ranges only from 1,900° to 2,200° F., while that of the highest-priced flint fire-brick is capable of withstanding a temperature of 3,000° F.

Occurrence of Flint Fire-Clays.—The flint fire-clays of Missouri occur in the eastern-central portion of the State, at a distance of from 40 to 140 miles west of St. Louis, along the Wabash, Colorado, or Rock Island, Missouri-Pacific and Frisco railroads. The clays are found in local pockets of very variable size that are possibly of Cretaceous age in limestones of Sub-Carboniferous and Lower Silurian age. The pockets are evidently old sink-holes in the limestone that have subsequently become filled with clay, and probably at such an extremely slow rate as to permit of the almost complete removal of the impurities by chemical solution, including some of the silica of the kaolinite base. The pockets are fringed with a sandy transition-bed that terminates in a sandstone which always flanks the sink-hole. This sandstone represents the silt or coarse matter that was promptly deposited, as a delta, by the inflowing streamlets that brought the detritus from the adjacent weathering limestone in rainy weather, while the finer, clayey matter was deposited in the central portion of the sink-hole. The sandstone always dips on all sides towards the center of the pocket, at angles of from 15° to 35°. The laminations of the clay-deposit have been destroyed by the chemical action of the water that filled the basin, so that the flint-clay has a very characteristic hard, massive, jointed structure, and breaks with a sharp conchoidal or flint-like fracture that gives the clay its name.

Most of the pockets are filled with a light-gray, exceptionally pure flint-clay that usually contains from 1 to 2.5 per cent. of impurities; occasionally it is more or less heavily contaminated with red iron oxide, which of course, ruins it for refractory purposes. The flint-clay has a hardness of from 3 to 3.5 and is devoid of plasticity, unless very finely ground; even then it is only slightly plastic. Even the weathered outcrop of flint-

clay is usually devoid of plasticity and consists of small, sharp, angular fragments.

The pockets resemble most sink-holes in being usually circular in shape and of very variable size, ranging from 25 to 300 ft. in diameter and from 10 to 40 ft. in depth. A sink-hole may produce from 5 to 500 car-loads before becoming exhausted. The flint-clay is usually worked on leases by local shippers, who pay the farm-owners a royalty of about 5 cents per ton. The pockets are worked in open pits and, with a haul of from 0.5 to 3 miles over poor roads, the flint-clay usually sells for $1 per ton f.o.b. cars at local points, or from $1.50 to $2 per ton at St. Louis.

Chemical Properties of Flint Fire-Clays.—The chemical analysis of the Missouri flint-clays is especially interesting not only in being very low in impurities, but in having from 0.5 to 3.5 per cent. excess of alumina above the amount contained in pure kaolinite. According to Dana and other leading authorities, kaolinite, the base of all clays, contains: Silica, 46.3; alumina, 39.8; combined water, 13.9; and total, 100 per cent.

The flint-clays of Missouri vary in these three essential constituents, as follows: Silica, 40.8 to 48.; alumina, 37. to 43.2; combined water, 13.3 to 15. per cent.

The water also is apt to be abnormally high as compared with pure kaolinite.

The percentage of fluxing-impurities—iron oxide, lime, magnesia and the alkalies—ranged in nine average samples collected from the various shipping-points, as follows: Iron sesquioxide, 0.15 to 0.83; lime, 0.28 to 1.93; magnesia, trace to 0.58; alkalies, 0.2 to 1.2. The total ranged from 1. to 2.89 per cent.

Such remarkable freedom from fluxing-impurities is very exceptional, and as these analyses are based on average samples collected over the face of the entire clay-bank, they must not be confused with the misleading analyses of selected specimens that are frequently found in text-books and in general technical literature. The influence of the adjacent source of supply of the clays, or the limestone, is noticeable in forming the largest portion of the impurities.

This great purity, combined with a coarse grain and high density, the latter being also very important factors of refrac-

toriness,[2] is the reason why these flint-clays are the most refractory of all known clays and are able to withstand a temperature of 3,000° F. with safety.

The flint-clays not being plastic, it is necessary to mix them with a plastic or "bond" clay in order to mold them into bricks, and local St. Louis fire-clay is added, either raw or after washing, for the St. Louis market. Efforts made to bind the flint-clays with very finely ground flint-clay have met with sufficient success to produce a brick, but it is tender and quite incapable of withstanding abrasion or careless handling in transit.

On account of the large amount of water contained in flint-clays that is expelled at a red heat, the shrinkage is excessive on burning, so it is necessary to calcine the clay before using it in order to eliminate the heavy fire-shrinkage, which ranges from 10 to 14 per cent.

Occurrence of Plastic Fire-Clay.—The plastic fire-clays of Missouri occur at the base of the Coal Measures in the eastern portion of the State in two different basins, known respectively as the St. Louis and Mexico areas. The St. Louis basin is the extreme western fringe of the Illinois coal-basin, which overlaps into St. Louis and St. Charles counties in Missouri. The Mexico basin is the eastern fringe of the Coal Measures that cover central western Missouri, and which is quite distinct from the Illinois Coal Measures.

Occurrence of St. Louis Fire-Clay.—The St. Louis basin, though much smaller in size, is much more important than the Mexico basin. It covers the western portion of the city of St. Louis, a large part of St. Louis county, and a small part of St. Charles county. The entire area, including a few small detached outlying basins, is in the form of an irregular polygon, and aggregates about 170 sq. miles. While there are several sheets of clays and shales in the lower Coal Measures in this basin, only one, the St. Louis fire-clay seam, is refractory, or capable of withstanding a heat of 2,500° F. which I regard as the line of demarcation between the refractory and non-refractory clays (this temperature being that of a pure white heat). This seam, which rests on the ferruginous sandstone, or

[2] Clay-Deposits of Missouri, by H. A. Wheeler, *Missouri Geological Survey*, vol. xi., p. 133.

the basal member of the Coal Measures, is a sheet of typical Coal Measure clay varying from 4 to 40 ft. in thickness, usually from 6 to 8 feet. It is nearly horizontal, has a roof of from 1 to 22 in. of bituminous coal, and rests on a transition-bed of sandy, green clay that merges into the ferruginous sandstone.

The fire-clay outcrops at the base of the valleys, where it is entered by adits, but on the higher ground it is worked by shafts from 25 to 125 ft. deep.

The St. Louis fire-clay has a light- to dark-gray color, and is a hard, rock-like clay when freshly mined that rapidly disintegrates into a soft, tough, plastic paste on exposure to the weather or when ground with water. It is worked on the "room and single-entry" system, and mined by hand-picking a side-cut along the pillar, and then blasting to this cut with black powder. Formerly this work was done by contract at from 35 to 50 cents per cubic yard (equivalent to about 2 tons), but it is now paid for by day-work, the miners receiving about $2 per day. While most of the fire-brick plants own their own clay-mines, there are a number of independent leasers operating on a royalty basis of from 3 to 5 cents a ton, who sell their output to St. Louis and to distant consumers. The principal mines are in the western part of St. Louis in what is known as the Cheltenham district, and the St. Louis fire-clay sells for from 60 cents to $1 a ton f.o.b. cars for mine-run, and from $1.50 to $2 for selected clay. These prices are about one-half the prices of New Jersey fire-clays.

There is a specially selected grade known as "pot-clay," used for making glass-pots, which is obtained from a purer, more uniform seam that occasionally occurs from 1 to 4 ft. thick in the middle or upper part of the bed. This pot-clay sells for from $10 to $14 a ton f.o.b. cars St. Louis.

The St. Louis fire-clay bed varies considerably in thickness, character and value. While the general character of the clay is coarse-grained and moderately siliceous, the silica ranging from 58 to 63 per cent., it occasionally becomes very coarse and sandy, and the silica increases to 70 per cent.; more frequently it becomes finer-grained, and the silica decreases to 50 per cent. The clay always contains more or less iron pyrites ("shiners," "sulphur" or "mundic"), which is usually disseminated throughout the clay in the form of fine crystals to the extent of

from 1 to 3 per cent., but at times it segregates into coarse granules and nodules. The latter form is exceptional, and while very easy to remove by washing, if allowed to get into the fire-brick it causes local fusion and a black slag. The redeeming feature of the excessively high percentage of iron in the St. Louis fire-clay lies in the fact that it is nearly always finely and uniformly disseminated through the clay—in fact, the pyrite crystals are usually too small to be removed by washing. It is the iron oxide that gives the brown color to fire-brick made from St. Louis clay when hard burned, while the fine pyrite crystals produce the small black specks. If underburned, the fire-bricks are from light-colored to white.

When the fire-clay occurs abnormally thick, as in the northern part of St. Louis county, the lower portions of the bed are more or less contaminated by calcareous concretions that resemble boulders of limestone; these ruin the clay for refractory uses, unless removed by careful sorting and washing.

Physical Properties of the St. Louis Fire-Clay.—The St. Louis fire-clay, after weathering or grinding, is more or less plastic. As measured by the tensile-strength test, it ranges from 80 to 150 lb. per sq. in. in air-dried briquettes, and the prolonged weathering of several years develops as high as 250 pounds. A fire-clay of average plasticity shows a tensile-strength of about 100 lb. per square inch.

When made up into a stiff mud with from 14 to 20 per cent. of water, the clay shrinks from 6 to 9 per cent. in air-drying; the air-dried clay shrinks from 4 to 8.5 per cent. when burned to vitrification, which is the limiting-point of shrinkage; and the total shrinkage varies from 11 to 16 per cent. The clay can be rapidly dried without cracking, and more or less rapidly heated without checking.

Incipient vitrification begins at from 2,100° to 2,250° F.; complete vitrification, or the safety-limit, occurs at from 2,300° to 2,450° F.; and viscosity, or initial fusion, takes place at from 2,500° to 2,700° F.; hence the clay is only moderately refractory.

The hard-burned ware forms a close body of great strength, which largely contributes to its very high value for zinc-retorts and glass-pots.

The St. Louis fire-clay is coarse-grained and has a high specific gravity, ranging from 2.40 to 2.47 in the mine-run, and

from 1.92 to 2.13 in the washed clay. It is largely this coarse grain and high density that renders the St. Louis fire-clay as refractory as it is, with its unusually high percentage of fluxing-impurities (for a fire-clay). In fact, extensive work on the Missouri clays, using the Le Chatelier pyrometer for the heat-measurements, has clearly brought out the fact that it is impossible to determine even approximately the refractoriness of a clay from the chemical analysis, because density and size of grain are found to have such important influence on the fusibility.

Chemical Properties of the St. Louis Fire-Clays.—The St. Louis fire-clay is most interesting in its chemical composition, for the reason that its high percentage of fluxing-impurities, combined with its high silica, would cause it to be unqualifiedly condemned for a refractory clay by the analytical chemist. As a matter of fact, it is unequaled in the United States for its superiority in making glass-pots, zinc-retorts and gas-retorts, while it makes an unusually durable fire-brick when the heat is not excessive. These statements allude to ware made exclusively of St. Louis fire-clay, without the addition of Missouri flint-clay to improve its refractoriness.

The fluxing-impurities—iron oxide, lime, magnesia and alkalies—usually range from 5 to 6 per cent., seldom as low as 4.5 per cent. and occasionally as high as 7.3 per cent. Good fire-clays are usually stated in published analysis to range from 2 to 4 per cent. of impurities, and as a 1-per cent. increase in the quantity of impurities present frequently lowers the fusibility at least 100° F., the chemist would be disposed to condemn all the St. Louis fire-clays for refractory requirements of even moderate demands. A high content of silica, which is always due to more or less free silica when it exceeds 46 per cent., also adds to the fusibility of a clay to a small but undetermined extent. Formerly it was supposed to be a harmless diluent, like titanic acid, but the researches of Seger seem to leave no doubt of the detrimental action of free silica in a refractory clay, though probably not to the extent he attributes.

In considering the preceding and subsequent analytical results, it is important to remember that the Missouri analyses were made on samples that were carefully taken over the entire clay-face or dump by disinterested experts, and are therefore actual averages of working-samples, whereas many quoted analy-

sis are on selected specimens that have too frequently been taken by interested parties.

The data given in Table III., based on air-dried average samples, gives the variations and average composition of the St. Louis fire-clay as obtained from seven different mines and four different washing-plants.

TABLE III.—*Composition of St. Louis Fire-Clays.*

	Mine-Run.		Washed.	
	Per Cent.	Average Per Cent.	Per Cent.	Average Per Cent.
Combined silica (SiO_2)	24.6 to 34.4	32.0	26.0 to 35.5	32.0
Free silica (SiO_2)	19.5 to 42.9	30.0	20.7 to 37.3	25.0
Alumina (Al_2O_3)	19.3 to 30.8	24.0	21.1 to 28.8	24.0
Combined water (H_2O)	7.7 to 11.9	10.5	8.9 to 11.4	10.0
Titanic oxide	trace to 2.0	1.0	trace to 1.9	1.1
Iron sesquioxide (Fe_2O_3)	0.8 to 4.7	1.9	1.8 to 2.5	1.85
Iron protoxide (FeO)	0.4 to 1.8	1.2	0.8 to 1.2	1.0
Lime (CaO)	0.4 to 1.2	0.7	0.6 to 0.9	0.7
Magnesia (MgO)	0.1 to 0.9	0.3	0.07 to 0.8	0.2
Potash (K_2O)	0.2 to 0.9	0.5	0.5 to 0.6	0.55
Soda (Na_2O)	0.2 to 0.6	0.2	0.1 to 0.4	0.10
Sulphur (S)	0.1 to 0.7	0.3	0.1 to 0.3	0.18
Sulphuric anhydride (SO_3)	0.1 to 0.6	0.35	0.2 to 0.6	0.4
Moisture	2.0 to 4.1	2.7	3.0 to 4.5	3.0
Fluxing impurities	4.3 to 7.3	5.5	4.7 to 5.2	4.8
Specific gravity	2.40 to 2.47	2.44	1.92 to 2.13	2.0

THE ST. LOUIS FIRE-BRICK INDUSTRY.

There are seven plants in St. Louis, and one in the adjoining St. Louis county, engaged in the manufacture of fire-brick and the usual tiles and special shapes called for by the refractory trade. Three make also gas-retorts that are shipped throughout the United States from the Atlantic to the Pacific coast. The plant in St. Louis county is only 5 years old, but the others are from 20 to 50 years old.

About $2,500,000 are invested in these plants and the annual value of the output in 1903 approximated $1,400,000, which is exclusive of a large output of sewer-pipe produced by two of the larger plants. These figures, while considerably larger than those given in the clay-statistics published annually by the U. S. Geological Survey, are regarded as more reliable; and, in addition, from $125,000 to $150,000 worth of fire-clay is produced annually by independent leasers, most of which is shipped to nearly all of the zinc-smelters in the United States, who depend on the St. Louis fire-clay for their zinc-retorts.

The Mexico, Mo., Fire-Clay.

The extent of the Mexico fire-clay basin has not yet been defined, and the workable fire-clay has only been found along the fringe, or at or near the outcrop, of the basal Coal Measures. It contains only one bed of refractory clay which has been worked at Fulton, Mexico and Vandalia, or over a distance of 50 miles, while outliers are known to occur beyond these limits. The deposit may be said to be an irregular parallelogram at least 75 miles long by from 5 to 15 miles wide.

The fire-clay rests on the ferruginous sandstone, with an intermediate transition-bed of sandy clay more or less stained green by iron. The seam is usually thick, varying from 6 to 40 ft., but only the lower 6 to 12 ft. of it is sufficiently pure for use as a refractory material. It is mined through shafts ranging from 40 to 150 ft. deep.

The clay is light- to dark-gray, hard and rock-like, and is blasted by the room-and-entry system. Considerable timbering, however, is required to support the roof.

Physical Properties.—The clay, after weathering or grinding, makes a fairly plastic paste that has a tensile-strength of from 40 to 80 lb. per sq. in. when dried. When molded with from 14 to 16 per cent. of water, it has a shrinkage of from 4 to 5 per cent., and an additional shrinkage when burnt to vitrification of from 6 to 7 per cent., giving a total shrinkage of from 10 to 12 per cent.

Incipient vitrification begins at from 2,200° to 2,300° F., complete vitrification, or the danger-line, occurs at from 2,400° to 2,500° F., while viscosity, or initial fusion, takes place at from 2,600° to 2,700° F.

Chemical Properties.—The fire-clay of the Mexico basin is much purer than that of the St. Louis seam, the proportion of iron oxide, especially, being much lower. The total percentage of fluxes range between the usual limits of most refractory clays.

The variation in composition is shown in Table IV. which is based on analyses of samples from the four principal mines in the Mexico district:

TABLE IV.—*Analyses of Clays of the Mexico District.*

	Per Cent.	Average Per Cent.
Silica,	47.0 to 55.0	52.0
Alumina,	30.0 to 37.0	33.0
Combined water,	10.5 to 13.0	12.0
Iron sesquioxide,	1.2 to 1.8	1.5
Lime,	0.4 to 0.7	0.5
Magnesia,	0.0 to 0.3	0.1
Alkalies,	0.5 to 1.4	0.7
Total fluxes,	2.5 to 4.2	3.4
Specific gravity,	2.43 to 2.46	2.44

THE FIRE-BRICK INDUSTRY OF THE MEXICO BASIN.

A plant of moderate size has been in operation at Vandalia and another at Fulton for about 20 years, and a third plant was operated at Mexico until 1895. In 1896 a small plant was built amid the flint-clays at Truesdale, but it had a short life. These plants have the disadvantage of no freight-competition; their coal costs considerably more than at St. Louis, and there is no home-market even for boiler-grade products. It is not surprising that this district has shown no growth in recent years in competition with the more favored St. Louis district. The capital invested aggregates about $150,000, and the annual value of the output is about $100,000.

The Zinc-Smelting Industry of the Middle West.

BY H. C. MEISTER, E.M., ST. LOUIS, MO.

(Lake Superior Meeting, September, 1904.)*

GENERAL HISTORY.

THE zinc-smelting industry of the United States has grown very rapidly in recent years and bids fair to outrival that of all other countries in the future. On account of the geographical situation of the chief ore-supplies, the industry may be divided into two main groups, the eastern and the western.

The eastern and older group includes the works of Penn-

* Presented at the supplementary session of the Institute in St. Louis, Mo.

sylvania, New Jersey and Virginia, which practically control their own ore-supply. The western group includes the works of Illinois, Missouri, Kansas, Wisconsin and Indiana, which do not control their own ore-supply, but are compelled to buy the ore in the open market. It is for this reason that the western zinc-smelting industry has been undergoing continual changes since its inception 40 years ago.

During the early discoveries of lead- and zinc-ore in the lead- and zinc-districts of Wisconsin and northern Illinois, the zinc-ore (smithsonite or "bonê" as the miner calls it) had no value and was considered waste; but later, when this ore was found in large quantities, it attracted the attention of metallurgists, which finally resulted, during the early sixties, in the building of the Matthiessen & Hegeler Zinc Works at LaSalle, Ill., the pioneer zinc-works of the West.

A few years later, zinc-ore was discovered associated with the lead-ore in Jefferson county, southeastern Missouri, which started the industry at Carondelet or South St. Louis. Still later, the discoveries of zinc-ore in the lead-ore region, first at Granby and then around Joplin, lead to the introduction of the industry in the southwest Missouri and Kansas districts.

At first the development of the industry was slow, but it progressed steadily and it is now one of the leading metallurgical industries of the middle West.

As this paper deals solely with the local zinc-industry the western group only is described. For convenience this group may be subdivided into States according to the progress of the development of the industry, which formerly placed Illinois at the head of the list, then Missouri, and later Kansas and Indiana. This order is now changed and Kansas heads the list, with Illinois second and Missouri third.

In a general way it may be said that after the establishment of the industry at LaSalle, Ill., in the '60's, there were four distinct periods of activity and enlargement followed by corresponding periods of depression. During the early '70's three plants were built at Carondelet, Mo., but after the panic of 1873, business became bad and did not revive until 1879. Activity was renewed in the beginning of the '80's, and many works were built in Kansas and southwestern Missouri. About 1885 a period of depression set in, followed by a very active

period of construction during the early '90's. After 1895 a period of dulness occurred, which lasted until 1898, when the activity was once more revived by the building of a number of large plants in the Kansas natural-gas district. Since that time development in this district has been so rapid that a shortage of ore soon occurred. In 1899 the ore-supply was insufficient, and the resultant keen competition caused the price of blende to advance to the unprecedented figure of $54 per ton. The gas-smelters having a strong advantage over the old coal-smelters in the cost of production, the severe competition soon caused the closing-down of all the coal-smelters in the Kansas and southwestern Missouri districts. The zinc-ores mined in connection with western silver- and gold-ores, however, soon relieved the shortage, and for the last four years large quantities of blende have been shipped from Colorado to Kansas for treatment in the natural-gas plants.

Production of Metallic Zinc in the United States.

The following statistics[1] of total output show the rapidity with which the production of spelter has increased, due largely to the application of natural gas.

	Short Tons.		Short Tons.
1895,	89,786	1900,	123,886
1896,	81,499	1901,	140,882
1897,	99,980	1902,	156,927
1898,	115,399	1903,	159,219
1899,	129,051		

The figures, given in the above table, include for each year about 10,000 short tons which represent the quantity made in the eastern and southern districts.

Roasting Zinc-Ore.

Most of the zinc-ore now mined in this district is zinc sulphide or blende, hence a preliminary roasting is necessary in order to get rid of the sulphur. The more thorough the elimination of the sulphur during the roasting, the better the chances are for the recovery of the zinc in the subsequent smelting-operation. Each unit of sulphur left in the roasted

[1] *Mineral Resources of the United States* for 1903, U. S. Geological Survey, p. 254.

ore will hold back double its weight of zinc, and as no sulphur by-products (such as matte) are formed in the smelting-operation, the loss by sulphur means so much waste in the residues which cannot be recovered. A variety of mechanical furnaces are used for roasting, although the old style hand-rabbled reverberatory furnaces are still in use at many plants.

The Hegeler Acid-Furnace.—Many years ago the Hegeler muffle-type of mechanically-rabbled furnace was introduced at the works of Matthiessen & Hegeler, LaSalle, Ill., with entirely successful results, but owing to the high cost of construction and the comparatively high cost of maintenance and operation, it has never been introduced except at works designed to make sulphuric acid from the sulphurous acid gases. At present the Hegeler furnace is used at the following works: Matthiessen & Hegeler Zinc Co., LaSalle, Ill.; Illinois Zinc Co., Peru, Ill.; United Zinc & Chemical Co., Iola, Kas.; and Grasselli Chemical Co., Cleveland, Ohio.

With the exception of a small plant at the works of the Mineral Point Zinc Co. using the contact-process for making sulphuric acid, the four companies above named are the only manufacturers of acid from blende in the United States, the chamber-process being in general use. The difficulty of finding a market for sulphuric acid in the West has no doubt prevented its more extended manufacture. The acid market, however, is gradually widening, and no doubt the manufacture of sulphuric acid from blende will become more common.

A new departure in this line of work has lately been made by the Grasselli Chemical Co., which manufactures sulphuric acid from Joplin blende at Cleveland, Ohio, and ships the roasted ore to West Virginia to be smelted with natural gas.

Mechanical Roasting Furnaces.—The roasting of zinc-blende in mechanically-stirred reverberatory furnaces marked the first advance in the reverberatory furnace practice. The first successful furnace of this type was the Brown horse-shoe furnace, invented in 1892, and its working gave an entirely new idea to inventors. Furnaces embodying the Brown invention sprang up like mushrooms all over the United States. Among the first to enter the field being the Pearce and the Ropp types, but litigation soon stopped their general introduction.

Compared with the ordinary hand-rabbled furnace, mechan-

ical furnaces effect a considerable saving in labor and yield a better and more uniform desulphurization of the ore roasted.

Mechanically-stirred furnaces of the Brown type are in use at the works of the Edgar Zinc Co., at Carondelet, Mo., and Cherryvale, Kas., and at the plant of the Collinsville Zinc Co. at Collinsville, Ill.

Recently, a number of mechanically-stirred furnaces of the Ropp type were in use at the plant of the Lanyon Zinc Co., Iola, Kas., but these had to be changed to the Cappeau type, in order to avoid an infringement of the Brown patents, the Ropp furnaces having been declared an infringement of Brown's patents.

SMELTING ZINC-ORE.

There has been no radical change in the method of smelting zinc-ore. The old style of distillation is still in use, all attempts at direct smelting having resulted in absolute failure.

The Belgian type of furnace is universally used throughout the middle West, either direct-fired or gas-fired, the latter with or without heat-recuperation.

Hegeler "Blow"-Furnace.—The first attempt at gas-firing was made by Matthiessen & Hegeler and resulted in the development of the Hegeler "blow"-furnace. Mr. Hegeler found that by firing with gas, the length of the furnace could be greatly increased, all that was necessary being to introduce the quantity of gas required to heat the whole furnace and burn it gradually by introducing air at intervals. The Hegeler furnace, as now built, consists of a long rectangular chamber having retorts placed four rows high, and in some cases six; the gases from the producers enter at one end and are discharged at the other. Air for combustion is introduced through iron pipes entering the front of the furnace at various places alongside of the retorts. Both air and gas enter under pressure produced by a positive blast. The producers are operated with a hot top and are placed close to the furnace. No recuperation or regeneration is attempted in the furnace, but the hot gases escaping are used for the generation of steam for power-purposes. These furnaces have been built in size varying from 224 to 504 retorts to the furnace, or from 448 to 1,008 retorts to the block.

Siemens Furnace.—The next attempt at gas-firing was made by the Illinois Zinc Co., at Peru, Ill. This company adopted the Siemens regenerative furnace, built on the well-known principle of the Siemens regenerative system. The Siemens producers, however, proved inefficient and were later replaced by steam-blast producers. The furnaces were originally built with large elliptical retorts, arranged in 3 rows, but lately they have been replaced with circular retorts placed 4 rows high. The latest improvement on these furnaces has been the Neureuther patent, which consists in extending the air- and gas-channels partly up into the center wall, by which both air and gas are admitted into the furnace-chamber at a higher point than was usual. By this means it is claimed that the furnace can be built 5 rows high without detracting from its efficiency. It is also claimed that the furnace is heated more evenly on account of the shorter reach of the flame.

In 1882 the Rich Hill Zinc Works at Rich Hill, Mo., and the Granby Mining & Smelting Co., at Pittsburg, Kas., each built furnaces of the Siemens regeneration principle, which were failures, and were abandoned later. The Rich Hill furnace was built exactly like those of the Illinois Zinc Co., having large elliptical retorts, in three tiers, while the Granby furnace was built with small circular retorts arranged in four tiers.

The failure of the Rich Hill furnace was partly due to mismanagement and partly to faulty construction. From my own observation I can say that the elliptical retorts were entirely too large for the heavy Joplin ore, and the furnace could not be worked "dry" in the required time; consequently the percentage of yield was poor. The Siemens producers required the use of a better grade of fuel which, at that time, was at least five times as expensive as the slack coal burned in the direct-fired furnaces. The most serious mistake, however, was in the arrangement of the gas- and air-channels underneath the checker-work. These channels, not being accessible during the time the furnace was in operation, soon became clogged with zinc oxide which had escaped from leaky retorts. So serious was the interference that an irregularity in the working of the furnace could be detected a few months after the furnace had been put in operation. This clogging caused frequent shut-downs, and instead of having produced a furnace with long campaigns and small

repairs, it gave short campaigns and frequent and extensive repairs. In the Granby furnace the circular retorts were better adapted to the work, but the same defects of construction existed as in the Rich Hill furnace and the results were the same.

The high cost of construction of the regenerative furnace and the inability to use the common slack-coal of the district, together with the poor work accomplished on account of faulty construction, soon brought the improved furnace into bad repute; and so prejudicial was the result that no more gas-furnaces were constructed, everybody reverting to the old style of direct-fired furnaces.

Direct-Fired Furnace.—If it had not been for the local cheap slack-coal, other attempts at gas-firing would no doubt have been made, but slack-coal could not be burned to advantage in independent producers, on account of its inferior quality; hence the direct-fired furnace came into general favor. These furnaces were cheap to construct and gave, as a rule, good average results, although the consumption of coal was large and the labor was excessive.

The first direct-fired furnaces built in Kansas had square fronts, and contained 7 rows of 14 retorts each, which amounted to 98 retorts to the furnace, or 196 retorts to the block. Later, the size of the furnace was increased to 16 retorts in a row, set 7 high, corresponding to 112 to the furnace, or 224 to the block, which remained the accepted type until the building of coal-furnaces ceased on account of the extensive discovery of natural gas in Kansas. In 1892 the natural-gas district of Indiana attracted attention, and a plant was built at Marion, followed by one at Ingalls and one at Upland.

Natural-Gas Furnace.—The application of natural gas to the zinc-distillation furnace, however, was not as simple as was at first supposed, and the first attempts for this purpose caused a great deal of worry and anxiety to the operators. The natural-draft furnaces of Indiana, in which a mixture of air and gas, on the principle of the Bunsen burner, was introduced through openings in the front or back of the furnace direct into the furnace-chamber, worked very irregularly. With the narrow channels of the zinc-furnace, it seemed impossible to distribute and control the heat properly, and frequent "butchers" or

break-downs of the retorts occurred with a correspondingly heavy loss of metal. Consequently, when the industry was started in the natural-gas district of Kansas in 1897, the first thing that suggested itself was to improve the distillation-furnace and the method of using the gas. Natural draft was discarded in favor of positive blast. In fact, the Hegeler blow-furnace was copied as near as conditions permitted. In the original form of the natural-gas-fired distilling-furnace, the gas was introduced at one end, in the same manner as in the producer-gas-fired furnace, and the heat regulated by the introduction of air at various places. This arrangement, however, was not satisfactory, for the reason that natural gas, having a high calorific value, was not completely burned in the furnace by the air introduced for this purpose, and, as a result of the incomplete combustion, large quantities of carbon were deposited in the spaces between the retorts which clogged the furnace. In order to insure the complete combustion of the natural gas, and thereby overcome the deposition of carbon, more air was forced in, which, however, caused the furnace to become overheated. A careful study of the requirements indicated a change in the method of introducing the gas and air, so that they would enter jointly at various places through the front. A furnace was then designed which proved satisfactory, but the regulation of the heat of this form of furnace remained a difficult task, and, unless watched very closely, irregularities and frequent break-downs occur.

The peculiar heat-requirements of a zinc-furnace differ from most other metallurgical furnaces in so far that there is no large space in which to make use of radiant heat. On account of the narrow channels between the numerous rows of retorts the heat must be supplied almost entirely by conduction, which makes it necessary to surround the retorts continually with a flame having the required temperature. As the retorts are easily destroyed by over-heating, an excess of heat must be carefully avoided. The strong cutting-action of a blow-pipe flame, such as is produced in the blow-furnace, easily causes overheating and a consequent destruction of the retorts. It also tends to an early destruction of the lining of the furnace, causing frequent and expensive repairs.

The blow-furnace was first built in sizes of 5 rows of 60 re-

torts (300 retorts) to the side. Later, the combustion-chamber built at one end was discarded and the space was utilized by adding 20 additional retorts, so that now the accepted size is 5 rows of 64 retorts (320) to the furnace, or 640 to the block. Lately, the tendency has been to limit the number of rows to four, with a corresponding increase in length of the furnace in order to hold the same number of retorts, an arrangement which lengthens the furnace, and in some works they have been divided into two transverse sections, placed in tandem in the same building. By working both sections as one furnace, the same economy of labor is retained. While the labor-requirement on these large furnaces is considerably less than on the small direct-fired furnaces, it is also more trying. The men being more exposed to the heat, which causes considerable trouble at times during the hot summer months. In order to avoid the trouble, the Edgar Zinc Co., at Cherryvale, limited each furnace to 5 rows of 20 retorts (100 retorts), or 200 to the block. By placing three blocks in one building, each separated from the other by a 12-ft. space, and operating the three small furnaces as one large one, the same economy of labor is obtained without having the men exposed to the heat as much as they would be with the single large furnaces.

These furnaces are also constructed to operate with natural draft and have proven quite successful. By admitting air and gas some distance below the lower row of retorts, primary combustion takes place before the gases reach the furnace-chamber proper; and by admitting more air, a secondary combustion takes place in the furnace-chamber, causing a very uniform heat without the deposition of carbon. Thus the cutting-action of the blow-pipe flame is entirely avoided, and repairs have been reduced to a minimum, which allows a long campaign of operation.

Lately, the coal-fired smelting-plant at Girard was removed to Chanute, Kas., to be operated with natural gas. The furnaces are arranged similar to those at Cherryvale, but each furnace has only 80 retorts, or 160 to the block, and the four blocks are placed in one building with each side operated as one furnace. These furnaces are also run by natural draft, similar to those at Cherryvale. Recently, a furnace was constructed at the works of the Prime Western Zinc Co., at Gas

City, Kas., using the recuperative principle of the Converse and De Saulles furnace. This furnace is a patented one which was developed at the plant of the New Jersey Zinc Co., at Palmerton, Pa.

The data in Table I., given on the following page, may not be strictly accurate, yet they show the changes that have taken place and the present condition of the zinc-industry.

Smelting in Colorado.—During the past few years the utilization of zinc-blende found associated with silver- and gold-ore in Colorado gave rise to the zinc-smelting industry at Pueblo, Colo. The United States Zinc Co., a branch of the American Smelting & Refining Co., built a plant at Pueblo, during 1903, which is said to be in successful operation now, although numerous obstacles were encountered at first, as is usually the case when opening up a new field.

Retorts for Zinc-Distillation.—Circular retorts, used exclusively in the West, vary but little in dimensions at the different works. They are usually about 10.5 in. in external and 8 in. in internal diameter and from 48 to 54 in. long. Formerly they were made by hand, but now they are almost exclusively made by machine, either the ordinary auger-machine or a hydraulic press. The latter type of machine is coming more into use and no doubt is the better of the two; it makes a very dense retort, which prevents leakage of zinc-vapor through its walls. A pressure of about 1,000 lb. per sq. in. is used. The capacity of both machines is about equal, but the cost of the hydraulic press plant is ten times that of the auger-machine.

Following the introduction of large furnaces, the method of operating has changed and numerous improvements have been made. Residues are now blown out from the retorts by a stream of water, instead of being scraped out with iron scrapers, as was formerly the practice.

Charging the Ore.—The ore-mixture is prepared outside of the furnace-building, instead of in front of the furnaces as formerly, and the charging is done in tiers instead of rows. The ore and coal are mixed mechanically and the mixture is placed in hopper-shaped charging-cars, from which it is charged into the furnaces. This mixture is usually prepared a day in advance, in order that it shall be ready for use when charging begins.

TABLE I.—*Data Relating to the Growth of the Zinc-Industry in the Middle West.*

Year of Construction.	Name of Company.	Abandoned. No. of Retorts.	Abandoned. Total.	In Operation. No. of Retorts.	In Operation. Total.
	COAL-BURNING SMELTERS—ILLINOIS.				
1863	Matthiessen & Hegeler Zinc Co., LaSalle			3,344	
1869	Illinois Zinc Co., Peru			3,350	
1875	Lumaghi Zinc Works, Collinsville	244			
1886	Collinsville Zinc Co., Collinsville			1,536	
1892	Waukegan Zinc Works, Waukegan			672	
1891	Winona Zinc Works, Winona			448	
1897	Sandoval Zinc Works, Sandoval		244	672	10,022
	MISSOURI.				
1869	Missouri Zinc Works, Carondelet	1,020			
1872	Martindale Zinc Works Edgar Zinc Co. Carondelet.			2,000	
1878	Carondelet Zinc Works				
1881	Joplin Zinc Works, Joplin	1,120			
1882	Rich Hill Zinc Works, Rich Hill			672	
1887	Robert Lanyon, Nevada	1,344			
1896	Geo. E. Nicholson, Nevada		3,484	672	3,344
	KANSAS.				
1874	Weir City Zinc Works, Weir City	1,200			
1878	Robert Lanyon, Pittsburg	1,200			
1881	S. H. Lanyon & Bro., Pittsburg	600			
1882	W. & J. Lanyon, Pittsburg	840			
1882	Granby Mining & Smelting Co., Pittsburg	912			
1888	Scammonville Zinc Works, Scammonville	400			
1886	Girard Zinc Works (old), Girard	448			
1890	Girard Zinc Works (new), Girard	896			
1891	Pittsburg & St. Louis, Pittsburg			896	
1890	Weir City North Works, Pittsburg			1,344	
1895	Chicopee Zinc Co., Cherokee	448			
1894	Midland Smelting Co., Bruce	1,120	8,064		2,240
	Totals		11,792		15,606
	NATURAL-GAS BURNING SMELTERS—INDIANA.				
1892	Columbia Zinc Works, Marion	800			
1893	Ingalls Zinc Works, Ingalls	744			
1893	Upland Zinc Works, Upland	600	2,144		
	KANSAS.				
1897	Lanyon Zinc Co. { Robert Lanyon Sons Spelter Co., Iola.				
1898	Lanyon Zinc Co. { Robert Lanyon Sons Spelter Co., La Harpe.			9,600	
1897	Lanyon Zinc Co. { W. & J. Lanyon, Iola.				
1898	Prime Western Zinc Co. { Prime Western Z. Co., Gas City.				
1902	Prime Western Zinc Co. { Nicholson Z. Wks., Iola.			8,900	
1902	Prime Western Zinc Co. { A. B. Cockerill, Gas City.				
1898	Cherokee-Lanyon Zinc Co., Gas City			2,400	
1898	Edgar Zinc Co., Cherryvale			4,800	
1901	United Zinc & Chemical, Iola			2,400	
1902	Granby Mining & S. Co., Neodesha			2,400	
1903	La Harpe Zinc Co., La Harpe			2,100	
1900	Chanute Zinc Co., Chanute			1,280	
1904	A. B. Cockerill, Altoona			1,800	
1904	Wm. Lanyon, Caney			1,800	37,480
	Total in gas-district		2,144		37,480
	Total in coal-district		11,792		15,606
	Grand total		13,936		53,086

Drawing the Metal.—The metal is now drawn in large ladles holding from 300 to 400 lb., instead of the former smaller ones holding from 40 to 50 lb. These large ladles are suspended from cranes attached to trucks running on rails in front of the furnace and supplied with protecting-shields. In some cases the ladle is hung to a shield suspended from a single overhead-rail fastened to the furnace-front and traveling the whole length of the furnace.

At present the three sheet-zinc mills in the United States are in the western district. Matthiessen & Hegeler first rolled sheet-zinc in the West, followed by the Illinois Zinc Co.; both having sufficient capacity to roll their entire product of spelter into sheet. In 1902 the Lanyon Zinc Co. built a small mill at La Harpe, Kas., which has since been in constant operation.

Manufacture of Zinc Oxide.

While the manufacture of zinc oxide is confined principally to the eastern district, a few establishments are in the West. Generally the requirements for making zinc oxide cannot well be met in the West. In the first place, anthracite coal, which is scarce in the West, is necessary, and cannot be had except at prohibitory prices. Oxidized zinc-ores free from cadmium are also necessary. Ores of this character occur in Wisconsin, and the large plant of the Mineral Point Zinc Co., a branch of the New Jersey Zinc Co., has been erected at Mineral Point to treat these ores. A small plant, started in 1902 at West Plains, Mo., was soon absorbed by the New Jersey Zinc Co., and I believe is now out of commission. A third plant, that of the Ozark Zinc Oxide Co., built some years ago at Joplin, has so far remained independent, but difficulty has been experienced at times in getting proper ore-supplies. At present these are obtained from New Mexico.

The zinc oxide industry will, in all likelihood, be controlled for some time to come by the New Jersey Zinc Co., which has practically an unlimited supply of ore admirably adapted to this purpose and abundant anthracite coal near its plants.

The Application of Dry-Air Blast to the Manufacture of Iron.

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THE atmosphere, which plays such an important part in the manufacture of iron and steel, is the most variable element involved in its several processes; and particularly is this true of the blast-furnace process, which consumes air in large quantities. At no time since the blast-furnace became an important and widely used apparatus—even when it was operated in the most crude manner—have the variations in composition of the raw materials used been as frequent and as great as the variations in humidity of the atmosphere. Important improvements have been made in the blast-furnace and its accessories, such as the hot-blast stoves, the increase in size and change in the shape of the furnace, more efficient blowing-engines, the increased protection given to the bosh-walls, and the careful preparation of the raw material, all of which have exerted a pronounced influence on the furnace-operations from a metallurgical standpoint. But during the past eight years little advance has been made in this direction; the fuel-consumption has not diminished, nor has there been any material increase in production. Within that period, however, there has been witnessed the greatest development in appliances for the economical handling of material. So complete has been the work in this direction, that, except in isolated cases, in this country at least, a further extension does not hold out much promise of a satisfactory return on the investment required. It seemed that, with the exception of the gas-engine, we had about reached the limit, for like a strong wall, the atmosphere, with its humidity as variable to-day as when first blown into a primitive blast-furnace, appeared to stand as a barrier to

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further progress. In furnaces using ore from the Lake Superior district the raw material, amounting to about 7,200 lb. per ton of iron, varies in composition within 10 per cent. and is as uniform as human skill can make it; but the atmosphere, of which 11,700 lb. are consumed per ton of iron, varies in its content of moisture from 20 to 100 per cent., from day to day and often in the same day, thus rendering the process, even with the best appliances, an uncertain one, because dependent on the caprice of the atmosphere.

The desiccation of the air used in blast-furnaces in such a way as to reduce its moisture to a small quantity, and to keep it uniform, must of necessity contribute in a very marked degree toward the attainment of uniformity in the furnace-operations. The advantages from desiccation can be appreciated only after due consideration is given to the volume of air that is consumed per minute and the large amount of moisture which it contains. Managers of blast-furnaces are familiar with the chilling-effects produced in the hearth by a tuyere that is leaking, which immediately results in a deterioration in the grade of the iron; yet the quantity of water ordinarily entering the furnace under these conditions is not greatly in excess of the quantity carried in, like a steady stream, by the atmosphere, during a period of the average humid conditions prevailing in the summer-season in this country.

It has been deemed preferable in this communication to express the quantity of moisture, contained in the atmosphere as aqueous vapor, in grains of water per cubic foot of air, inasmuch as the quantity of air blown into blast-furnaces is expressed in cu. ft. With air containing 1 grain of water per cu. ft., there is passed into the furnace, for each 1,000 cu. ft. used per min., practically 1 gal. of water per hour. A furnace of average size in the Pittsburg district consumes about 40,000 cu. ft. of air per minute, which would pass into the furnace 40 gal. of water per hour for each grain of moisture contained in a cu. ft. of air. The quantity of moisture in the air, taken from daily readings by the observer of the United States Weather Bureau at Pittsburg, is set forth in Table I. on the following page.

TABLE I.—*Moisture in Air at Pittsburg. U. S. Weather Bureau.*

	Average Temperature.	Weight of Water per Cu. Ft. of Air.	Quantity of Water Entering per Hour into a Furnace Using 40,000 Cu. Ft. of Air per Minute.
	Degrees Fahrenheit.	Grains.	Gallons.
January,	37.0	2.18	87.2
February,	31.7	1.83	73.2
March,	47.0	3.40	136.0
April,	51.0	3.00	120.0
May,	61.6	4.80	192.0
June,	71.6	5.94	237.6
July,	76.2	5.60	224.0
August,	73.6	5.16	206.4
September,	70.4	5.68	227.2
October,	56.4	4.00	160.0
November,	40.4	2.35	94.0
December,	36.6	2.25	90.0

Table I., like all records made by the U. S. Weather Bureau, is from observations taken on the top of a high building, and does not correctly indicate the condition of the atmosphere at the furnaces where the air is used. In fact, at one of the steel-works in Pittsburg, observations made simultaneously at three separate stations showed a perceptible variation in moisture. For the purpose of comparison with observations of the U. S. Weather Bureau, there is shown in Table II. the average monthly content of moisture in the air at the furnaces, the observations being made at 9 a.m.

TABLE II.—*Moisture in Air at Steel-Works in Pittsburg.*

	Weight of Water per Cu. Ft. of Air.
	Grains.
January,	2.8
February,	2.7
March,	3.1
April,	3.3
May,	4.7
June,	7.3
July,	7.0
August,	7.1
September,	5.4
October,	3.2
November,	3.3
December,	3.0

The variations in moisture from month to month set forth clearly the conditions, as to atmosphere, with which blast-fur-

naces in this country have had to contend. If these conditions were uniform throughout the whole month, it would not be a difficult problem to deal with; but unfortunately they are not uniform, and it is instructive to note the changes which occur from day to day in the same month. In Table III. is shown a record worked out from data furnished by the U. S. Weather Bureau at Pittsburg. These observations represent a different period from that shown in Table I.; they were taken at 8 a.m. and 8 p.m., and show the grains of water per cubic foot of air at the time observed, for the months of January and July.

TABLE III.—*Moisture in Air at 8 a.m. and 8 p.m.*

	JANUARY. Quantity of Water.		JULY. Quantity of Water.	
	8 A.M.	8 P.M.	8 A.M.	8 P.M.
Day of Month.	Grains.	Grains.	Grains.	Grains.
1,	1.96	3.06	7.24	7.48
2,	2.55	3.66	8.23	7.98
3,	2.46	3.80	8.50	7.48
4,	2.07	2.27	8.50	7.48
5,	1.81	1.12	8.46	7.72
6,	0.99	1.12	6.50	8.24
7,	1.16	1.67	8.78	7.47
8,	1.49	1.88	7.98	7.24
9,	1.96	2.19	6.78	5.94
10,	1.81	1.88	7.48	6.35
11,	1.74	1.55	7.98	7.48
12,	1.55	1.07	6.73	6.35
13,	0.99	1.55	5.94	4.84
14,	1.61	1.81	5.55	5.74
15,	1.67	1.96	5.74	5.19
16,	2.04	2.27	6.35	6.35
17,	2.45	3.29	7.72	7.98
18,	1.81	1.32	7.24	7.24
19,	1.12	1.16	8.24	7.48
20,	1.43	2.11	7.48	7.24
21,	2.11	1.88	7.72	7.38
22,	1.88	1.88	6.78	5.74
23,	0.91	1.17	7.43	6.35
24,	0.99	2.11	6.56	6.11
25,	0.69	1.83	6.05	7.74
26,	0.61	0.99	7.72	7.32
27,	0.56	0.88	7.98	7.48
28,	0.72	0.70	6.56	5.74
29,	0.76	0.80	6.14	5.01
30,	0.95	1.12	5.74	6.35
31,	0.70	1.41	6.56	5.19

It will be observed from the data in Table III., that, while the moisture in the atmosphere in the month of January is much less than in July, yet the percentage of variation is greater. In order to illustrate more precisely the exact conditions with respect to the atmosphere, under which blast-furnaces must be operated, there is shown in Tables IV. and V. a record of observations taken each hour in the day. In order not to make the data too burdensome the months of April and October have been selected, as they represent months between the warm and cold seasons, and will also serve for comparison with January and July, as shown in Table III.

It should be stated with reference to the Tables IV. and V., that observations were taken with a stationary instrument, which shows results somewhat higher, and not as accurate, as those taken with a whirled psychrometer. Nevertheless, they were taken with the same instrument and are relatively correct. By simply multiplying the grains of moisture by 40—which represents the number of gallons of water entering a modern furnace per hour, for a content of 1 grain of moisture in a cubic foot of air—a clear idea can be had of the gallons of water entering the furnace per hour, for the various conditions of humidity. The changes are great not only from day to day, but from hour to hour in the same day, and often they are very abrupt. These records were made at a furnace-plant, situated on the bank of a river, where the conditions exist for an increased humidity as compared with higher ground; and to what extent the abrupt changes may have been caused by the presence of steam in the atmosphere—absorbed from spraying the hot pig-beds, the blow-off from boilers and exhaust from engines, or from a rain-storm, when the humidity decreases suddenly—it is impossible to say. How frequently has it happened in the experience of every furnace-manager, that the furnace has gradually or suddenly lost its hearth-temperature and produced a grade of iron either undesirable or unmarketable, without any visible cause. Tuyeres are examined for leaks, the raw material in the stock-yard is carefully inspected, and usually the coke is condemned. A more intimate acquaintance with the atmosphere would have provided a correct and ready reason, for the variations therein are not only many times greater than in the raw material, but a greater weight of it is used per ton of iron produced.

It is true that the atmosphere has been recognized by numerous metallurgists as the cause of many serious irregularities in blast-furnace operations, but it is doubtful whether its influence has been adequately recognized. Many writers on metallurgical subjects have considered the moisture in the atmosphere, and calculated—and invariably underestimated—the absorption of heat necessary for its dissipation; and have dismissed the subject with the conclusion that to extract the moisture the game was not worth the candle, or in a spirit of resignation accepted it—like storm and sunshine—as a condition beyond our control. This conclusion has no doubt been reached by a consideration alone of the quantity of fuel necessary to dissipate the moisture in the furnace-hearth, based on observations of the humidity of the atmosphere taken outside the blowing-engine room; and this quantity, while important, does not indicate a great saving in fuel. Of much greater importance is the variation in moisture from time to time and the margin of heat carried in the furnace to compensate for these variations, which margin is invariably large; and every furnace manager is aware of its existence, from the way in which he is required to manipulate the hot-blast temperatures, and from the silicon in the metal, which is the thermometer of the hearth.

It has often been a matter of surprise that a greater saving of fuel per ton of iron was not obtained in winter, as compared with summer season, since the winter records show much less moisture in the atmosphere. The reason is, that blowing-engines at blast-furnaces do not receive air of the dryness shown in the Tables I. to V. In summer the windows and doors of the blowing-engine room are wide open, and the humidity of the air-supply is practically that of the atmosphere; but in winter they are nearly or quite closed, and the entering-air has mixed with it all of the steam that leaks from the engine. Records taken over a number of years show that there is not a very great difference in the moisture in atmosphere between observations taken outdoors in summer and in the engine-room in winter. In Table VI. are monthly records showing a comparison between winter and summer months, the observations having been taken indoors and outdoors respectively.

TABLE IV.—*Hourly Record of Moisture in Air.*

Grains of moisture per cubic foot of air.

April	A.M.							P.M.												A.M.				
	6	7	8	9	10	11	12	1	2	3	4	5	6	7	8	9	10	11	12	1	2	3	4	5
1	2.77	3.30	3.71	3.71	3.09	3.46	3.46	3.46	3.46	3.46	3.46	4.11	3.71	3.71	3.71	3.30	3.71	3.71	3.71	3.30	3.71	3.71	3.71	3.09
2	3.71	3.71	3.30	3.30	3.71	3.71	3.46	3.46	3.46	3.81	3.46	3.81	3.09	3.81	3.30	3.77	2.77	3.71	3.71	3.71	3.71	3.71	3.68	3.30
3	3.09	3.09	3.09	3.09	3.46	3.46	3.46	3.09	3.09	3.46	3.09	3.30	3.30	3.02	3.02	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30
4	3.71	3.71	3.71	3.71	3.71	4.11	4.11	4.67	4.67	3.81	3.81	4.11	3.30	4.01	3.30	3.71	4.01	3.71	3.71	3.30	3.38	3.30	3.30	4.01
5	3.68	3.68	4.01	3.30	4.01	3.89	3.71	4.11	4.11	3.46	4.11	4.11	4.11	3.46	3.09	3.09	3.09	2.47	2.77	2.77	2.77	2.77	2.77	2.77
6	2.47	3.09	3.09	3.09	3.71	3.71	3.71	3.71	3.46	3.46	3.71	3.71	3.71	3.71	3.71	3.71	3.09	3.30	3.30	3.30	2.77	2.77	2.77	2.47
7	2.77	3.30	3.30	2.77	4.01	3.30	3.71	3.81	4.11	4.33	4.67	4.67	4.11	4.01	3.30	3.30	3.71	3.71	3.30	3.30	3.30	3.30	3.30	3.30
8	3.02	3.30	4.01	3.30	3.71	3.71	3.71	4.11	4.11	4.40	4.40	4.40	3.71	3.71	4.01	4.01	3.30	3.30	3.30	3.30	3.30	3.02	3.02	3.30
9	3.68	3.68	3.71	4.11	4.11	4.11	4.28	4.67	4.01	4.01	4.31	4.31	3.68	3.68	3.68	4.01	3.68	3.68	3.30	3.39	3.39	3.02	3.39	3.02
10	3.68	3.68	3.98	3.68	3.68	4.01	3.71	3.71	4.67	4.17	5.11	5.46	5.11	5.35	5.11	5.14	5.35	5.20	4.75	4.40	4.75	4.40	4.75	4.75
11	4.75	4.31	4.40	5.01	4.33	5.28	4.56	4.56	5.22	5.22	5.22	5.22	4.24	4.96	3.81	3.81	3.46	3.46	3.81	3.81	3.46	3.46	3.46	3.46
12	3.46	3.46	3.30	3.46	3.98	3.30	3.84	3.68	3.47	3.20	3.52	3.52	3.17	3.11	2.74	3.98	3.46	3.71	3.71	3.09	3.09	3.30	3.30	3.30
13	3.02	3.68	3.71	4.01	4.96	4.71	5.22	4.85	4.59	4.85	4.56	4.56	4.56	4.96	4.96	4.67	4.81	5.01	5.01	4.75	4.75	4.31	4.31	4.31
14	3.68	3.68	3.68	3.68	4.31	4.01	4.01	4.01	4.01	4.31	4.31	4.40	4.31	4.31	4.31	4.31	4.31	3.98	4.31	3.68	3.98	4.31	3.98	3.98
15	4.01	4.01	3.71	4.40	4.41	4.75	4.33	4.96	4.56	4.24	4.24	4.24	4.33	3.81	4.11	3.81	3.46	3.30	3.30	3.30	3.30	3.68	3.62	3.62
16	3.68	3.68	4.01	3.81	3.81	4.67	3.52	4.96	4.96	4.96	4.96	4.96	4.96	5.01	4.40	4.40	4.11	4.75	4.46	4.67	5.01	5.01	5.01	5.01
17	5.01	4.67	5.60	5.26	5.28	5.97	6.27	6.27	6.27	6.27	5.83	6.27	6.25	5.60	5.60	5.60	5.35	5.35	5.77	4.40	4.75	4.75	4.75	4.52
18	4.75	4.75	5.35	5.35	5.35	5.32	5.60	6.08	6.43	6.43	6.62	5.84	5.84	5.84	5.77	5.77	5.11	5.11	5.77	5.77	5.77	5.35	5.77	5.77
19	5.77	5.35	5.77	5.77	6.32	5.77	6.32	6.32	6.32	6.32	6.32	5.84	6.72	5.77	4.75	4.75	4.75	4.01	4.01	4.31	3.30	3.30	3.30	4.01
20	4.01	3.30	3.30	3.30	3.30	3.30	3.30	3.71	4.40	4.40	4.40	4.40	3.71	3.30	3.30	3.30	3.30	3.30	3.09	3.30	3.30	3.30	3.30	3.30
21	4.01	4.40	4.40	3.71	3.71	4.40	4.11	4.11	4.11	4.67	5.01	5.01	5.35	4.01	4.01	4.01	5.11	4.75	4.31	4.01	4.01	4.31	5.11	4.31
22	3.30	4.01	3.46	3.71	4.11	4.11	4.33	4.56	4.98	5.22	5.59	5.10	4.24	4.24	4.56	4.24	5.22	5.22	5.22	5.59	5.59	5.28	5.01	4.40
23	5.35	5.01	5.01	5.60	5.28	5.28	5.84	5.35	6.18	6.18	6.18	6.18	6.18	6.39	6.39	6.18	6.18	6.18	5.77	6.18	6.18	6.18	5.11	5.11
24	5.46	5.11	5.11	5.11	5.11	5.11	5.11	5.11	4.75	5.11	4.75	5.77	4.75	5.11	5.11	5.46	5.77	5.77	5.35	4.75	4.75	5.11	5.11	4.75
25	4.31	4.41	4.75	4.75	4.40	4.40	4.40	5.01	5.01	5.01	5.01	5.01	4.75	4.75	4.40	4.40	3.71	3.75	4.05	4.00	4.01	4.01	4.01	4.01
26	4.01	4.40	4.11	4.11	4.61	4.67	4.67	4.67	4.67	4.67	4.33	4.33	4.33	4.33	3.71	4.01	4.31	4.01	3.68	4.01	3.30	3.68	3.68	3.68
27	4.31	4.01	4.40	4.11	4.67	4.67	4.96	4.33	4.33	5.28	4.33	4.96	4.67	3.71	4.01	4.40	4.01	4.01	4.01	4.01	3.68	3.68	3.68	3.68
28	3.30	4.01	3.46	3.81	4.11	4.11	4.67	4.67	4.67	5.28	5.28	4.33	5.01	4.40	4.11	4.40	4.40	4.40	3.71	4.31	4.01	4.01	3.71	4.40
29	4.01	4.01	4.40	4.40	4.40	4.11	4.11	3.81	4.40	4.40	4.11	4.40	4.40	4.01	4.31	4.75	4.75	4.07	4.01	4.01	4.01	4.01	4.31	4.01
30	4.01	3.68	4.31	3.81	4.67	4.33	5.60	5.28	5.97	6.43	5.28	5.28	5.84	5.01	5.35	5.77	4.75	4.75	4.75	5.11	4.75	4.75	4.75	4.75

TABLE V.—*Hourly Record of Moisture in Air (Continued).*

Grains of moisture per cubic foot of air.

October	A.M.							P.M.												A.M.				
	6	7	8	9	10	11	12	1	2	3	4	5	6	7	8	9	10	11	12	1	2	3	4	5
1	5.46	6.18	7.08	6.62	6.27	6.27	6.64	6.39	6.39	7.73	7.73	7.65	8.15	7.65	7.16	7.08	6.32	7.08	6.32	6.72	6.72	7.20	6.18	6.18
2	7.20	7.92	8.86	9.38	9.02	10.15	10.15	10.15	10.10	10.15	10.15	9.38	10.02	10.02	8.86	8.86	8.86	7.92	7.92	8.86	8.40	8.40	8.40	7.20
3	7.92	7.92	9.38	9.02	9.57	9.57	9.13	9.78	9.78	9.14	10.89	10.15	8.54	8.54	8.54	8.86	7.92	7.92	7.64	8.86	8.54	9.51	7.92	9.05
4	9.05	7.92	8.54	9.02	10.15	10.15	10.15	9.02	10.02	10.02	10.27	10.02	9.02	9.38	9.38	9.02	8.86	8.86	9.05	9.84	9.51	9.84	8.40	9.84
5	8.40	9.51	9.51	8.40	9.51	9.51	9.51	8.86	8.86	8.54	8.54	9.51	7.64	7.64	7.64	7.92	8.40	6.72	7.92	6.72	5.77	5.77	5.77	5.35
6	5.35	5.35	5.35	5.01	5.60	5.28	4.96	5.59	5.22	5.22	5.59	5.28	5.01	5.35	5.11	5.11	5.11	5.11	5.11	5.46	5.99	5.11	5.77	5.11
7	6.18	6.18	5.77	5.35	6.32	6.32	7.64	7.92	7.20	8.40	7.92	7.92	7.52	7.20	7.20	7.20	7.20	7.20	7.20	7.20	7.52	7.52	7.52	6.18
8	7.20	7.20	7.20	6.72	6.72	5.60	5.28	6.43	6.43	5.81	6.32	6.72	6.72	6.72	5.77	5.77	5.77	5.77	6.18	5.77	6.18	5.11	6.18	5.11
9	5.46	5.46	6.18	5.35	4.67	4.56	4.85	5.22	4.85	5.22	4.56	5.77	4.40	5.11	5.11	5.11	4.31	4.31	4.40	5.11	5.11	4.31	4.66	4.66
10	4.66	5.11	4.75	4.75	5.77	6.62	7.47	7.47	8.15	7.25	6.71	7.47	7.08	7.08	7.08	7.47	7.25	6.71	6.71	6.43	6.43	5.60	5.84	.....
11	6.32	6.43	6.71	6.27	5.83	6.39	6.71	5.97	5.35	4.75	4.75	4.75	6.72	5.77	5.77	5.77	5.77	6.18	6.18	4.66	4.31	5.11	5.11	4.31
12	4.31	3.75	4.01	3.81	4.33	3.98	3.71	3.71	3.71	3.98	3.98	3.52	3.52	3.82	3.81	4.01	4.01	4.01	4.31	4.31	4.31	4.31	4.31	4.31
13	4.31	4.31	4.01	5.01	5.28	5.28	4.85	4.85	4.47	4.24	4.24	4.56	4.67	4.67	4.67	4.75	5.35	4.75	5.11	4.75	4.75	4.40	3.81	3.46
14	3.11	3.11	3.46	3.11	3.11	3.11	3.11	2.75	3.46	3.71	3.71	3.71	3.71	3.71	3.71	4.01	4.01	3.71	4.11	4.11	4.11	4.11	3.46	3.46
15	3.81	3.46	3.46	3.11	3.11	3.81	3.81	3.52	3.98	3.52	3.52	3.11	3.81	3.11	4.01	4.01	3.30	3.30	3.68	3.68	3.68	3.68	3.68	3.68
16	3.68	3.68	3.68	4.01	4.11	4.33	4.33	4.33	3.98	3.98	5.01	3.71	4.40	4.01	4.01	4.01	4.01	4.01	4.01	4.01	3.71	3.71	3.71	3.71
17	3.71	3.71	4.40	3.81	3.30	3.84	4.01	4.01	4.01	4.47	4.47	4.47	3.52	3.84	3.84	3.30	4.24	3.30	3.71	4.54	6.67	4.67	4.33	3.81
18	4.40	4.75	4.75	4.75	4.75	4.75	4.75	5.11	4.75	4.75	4.75	4.11	4.11	4.40	4.11	4.40	4.11	4.40	3.71	3.71	3.71	3.71	4.01	3.71
19	3.71	3.68	4.01	4.11	3.98	3.71	3.30	2.95	2.95	3.84	3.84	3.98	3.98	3.81	4.11	3.71	3.46	3.71	4.01	4.01	4.01	3.68	4.01	4.01
20	3.68	3.68	4.01	4.01	4.33	4.56	3.84	4.85	3.82	4.01	4.24	4.24	4.67	4.11	4.11	4.40	4.40	4.40	4.11	4.11	5.01	4.40	4.75	4.75
21	5.77	5.35	5.77	5.77	6.18	5.77	5.77	5.77	5.77	5.35	6.32	5.01	5.35	5.35	5.77	6.72	5.35	5.35	5.77	5.77	4.40	4.40	4.40	3.46
22	3.46	3.46	3.46	3.46	3.46	3.11	3.46	3.46	3.46	3.46	3.46	3.52	3.81	3.81	3.52	3.11	3.11	3.81	3.81	3.81	3.81	3.81	3.81	3.81
23	3.52	3.52	3.17	3.52	2.75	3.52	3.17	3.17	3.52	3.17	3.52	3.11	3.52	3.52	3.11	3.11	3.46	3.46	3.46	3.09	3.71	3.09	3.30	3.30
24	3.30	3.30	3.71	3.46	4.33	3.98	4.96	4.56	4.24	4.24	4.56	4.67	3.81	4.11	4.11	4.11	4.40	3.71	3.71	3.71	3.71	4.01	4.01	3.71
25	4.01	3.30	3.71	4.11	4.33	4.24	4.85	4.85	4.01	4.47	3.84	3.98	4.33	4.11	3.46	3.71	3.71	4.01	4.40	4.40	4.11	4.11	4.11	4.11
26	4.16	3.71	3.46	3.46	2.74	2.75	2.75	2.75	3.11	3.46	3.46	3.09	3.11	3.46	3.11	2.74	2.74	2.74	2.52	2.75	2.23	2.52	2.74	2.52
27	2.52	2.23	2.52	2.23	2.23	2.23	2.23	2.75	2.75	2.45	2.45	2.52	2.74	2.74	2.74	3.09	3.09	3.09	2.47	3.09	3.09	3.09	2.47	3.09
28	3.30	3.30	2.74	3.46	2.75	2.75	2.94	2.94	3.30	3.30	2.94	2.75	2.74	3.11	3.46	2.74	3.09	3.46	2.74	3.09	3.46	3.09	3.09	3.30
29	2.74	2.74	2.74	3.09	2.75	3.52	3.71	2.95	3.30	3.30	2.58	3.81	3.11	3.11	3.46	3.11	3.46	2.75	3.11	3.11	3.09	3.09	3.09	3.09
30	3.30	3.46	3.30	3.46	4.01	3.71	3.46	3.46	3.46	3.11	2.75	3.17	2.75	3.11	3.46	3.46	3.11	3.11	3.11	3.11	3.46	3.46	2.74	3.71
31	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.71	3.46	3.46	3.46	2.75	3.46	3.46	3.09	3.09	3.09	3.09	3.09	3.30	3.39	2.77	3.09	3.30

TABLE VI.—*Moisture in Air in Winter and in Summer.*

WINTER.		SUMMER.	
Month.	Quantity of Water per Cu. Ft. of Air.	Month.	Quantity of Water per Cu. Ft. of Air.
	Grains.		Grains.
January, . . .	4.5	April, ...	4.2
February, . . .	4.6	May, ...	4.1
March, . . .	4.7	June, ...	6.4
October, . . .	6.4	July, ...	5.2
November, . . .	4.6	August, ...	6.7
December, . . .	5.0	September, . . .	5.7

From a comparison of Table VI. with Table II. it certainly appears that a material advantage could be gained by leading pipes from outdoors to the inlet-valves of the air-cylinder. So impressed did I become with this conclusion, that the blowing-engines at a furnace under my direction were thus equipped in the month of January, and continued to draw the supply of air from outdoors throughout the year. The excellent results expected in the winter season did not materialize, or rather, were so slight, as compared with a companion furnace not so equipped, as to argue against any further experiment in that line. This experience suggested the conclusion that while the air in the engine-room was higher in its content of moisture, through its admixture with steam, than the outside air, yet it was not subject to the same variations; and further that these variations, which were often sudden and great, were really the most troublesome feature, and that nothing less than maintaining the atmosphere uniform with respect to humidity would prove of any material advantage. The saving in fuel through such uniformity could not be accurately set forth. The amount of fuel necessary for the decomposition of the moisture in the blast could be closely arrived at; but to what extent the reserve of heat, needed for counteracting the variations in moisture, could be diminished by desiccation of the blast, could only be approximately estimated, in the absence of definite data. Nevertheless, the possible saving in that respect was deemed to be considerable.

Of course, a wide field of experiment had to be covered in order to determine the most feasible method and apparatus for extracting the moisture. Various schemes for its direct absorp-

tion were worked out and in turn abandoned; and refrigeration by means of anhydrous ammonia was finally chosen. After many preliminary experiments an insulated chamber containing coils of pipe, and of sufficient size to treat the air from a blowing-cylinder 3 ft. in diameter, was built. A small ice-machine was installed to circulate the ammonia through the coils, and the air was admitted to the refrigerating-chamber from an auxiliary chamber into which steam could be introduced at will, thus making it possible to treat, at any time, air containing the maximum amount of moisture with which it would be necessary to contend in the summer months. In this experimental plant air was treated under a variety of conditions for a considerable period, and from the data obtained the equipment for a modern furnace was worked out.

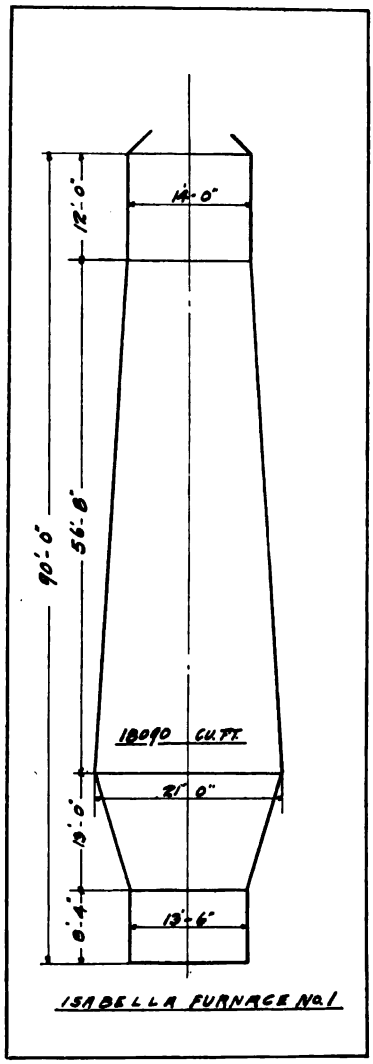
In the original plan, the refrigerating-chamber was placed between the blowing-engine and the furnace, so that the air passed through it under blast-pressure. At a furnace of modern type, this pressure may range from 15 to 30 lb. per sq. in. —more than double that which was in use when this process was first conceived. In the practical working-out of the process, it was found that workmen must frequently enter the refrigerating-chamber, to regulate the flow of the refrigerant through the pipes, and the thawing-off of the frost. A high pressure in the chamber would make its supervision and operation extremely difficult and hazardous. This consideration, together with that of the more expensive construction required by the high pressure of the passing air, led to the placing of the chamber so that the air should be drawn through it at atmospheric pressure to the blowing-engines.

The Isabella furnaces of the Carnegie Steel Co., situated at Etna, Pa., a suburb of Pittsburg, were selected as the plant at which to install the apparatus for applying the dry-air blast.

The lines and dimensions of this furnace, shown in Fig. 1, represent the usual construction of furnaces in the Pittsburg district. The furnace is blown with twelve 6-inch tuyeres, and is equipped with four hot-blast stoves. Blast is supplied by three blowing-engines having the following dimensions: Steam-cylinder, 44 in. in diameter; air-cylinder, 84 in.; stroke, 60 inches.

Fig. 2 shows, in elevation, the ammonia-compressors, condensers, and the refrigerating-chamber. This view of the re-

FIG. 1.



LINES AND DIMENSIONS OF ISABELLA FURNACE, No. 1.

frigerating-chamber shows it to be connected for the direct expansion of ammonia; but, as the escape of ammonia gas through a broken pipe or leaking-joint might imperil the life of anyone

in the chamber at the time, it was decided to adopt the brine system, and the pipe-connections are as shown in Figs. 3 and 4, representing the refrigerating-chamber in longitudinal and

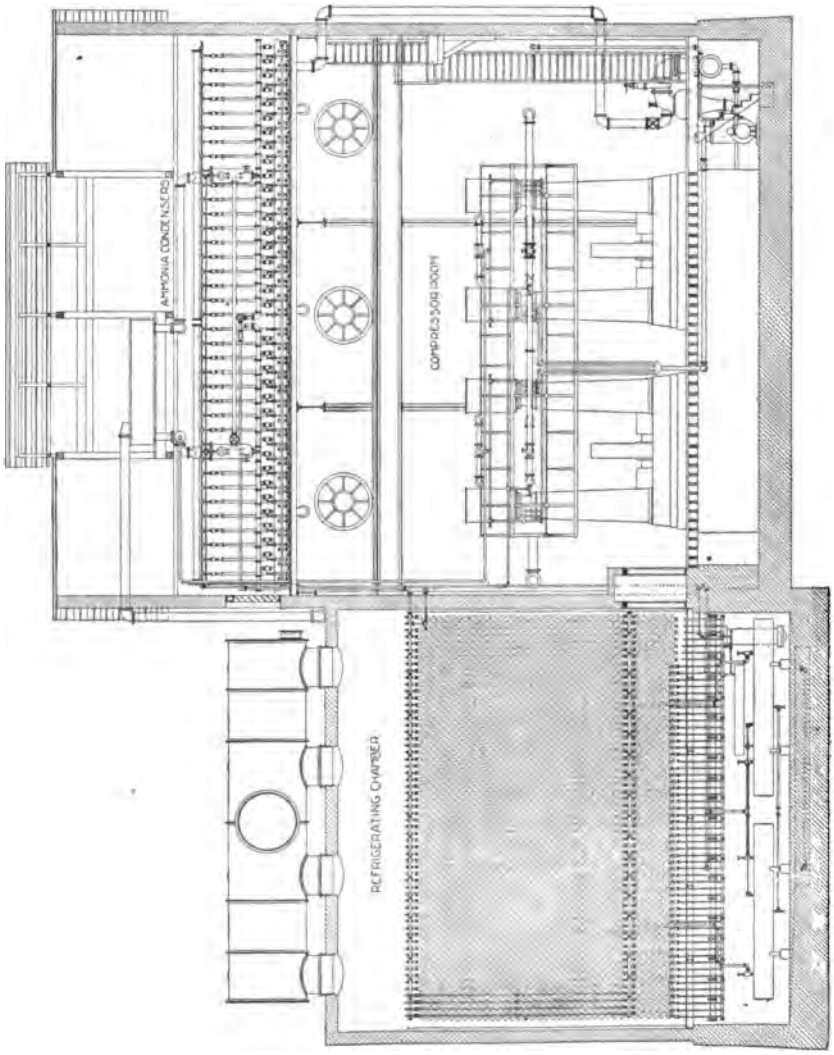


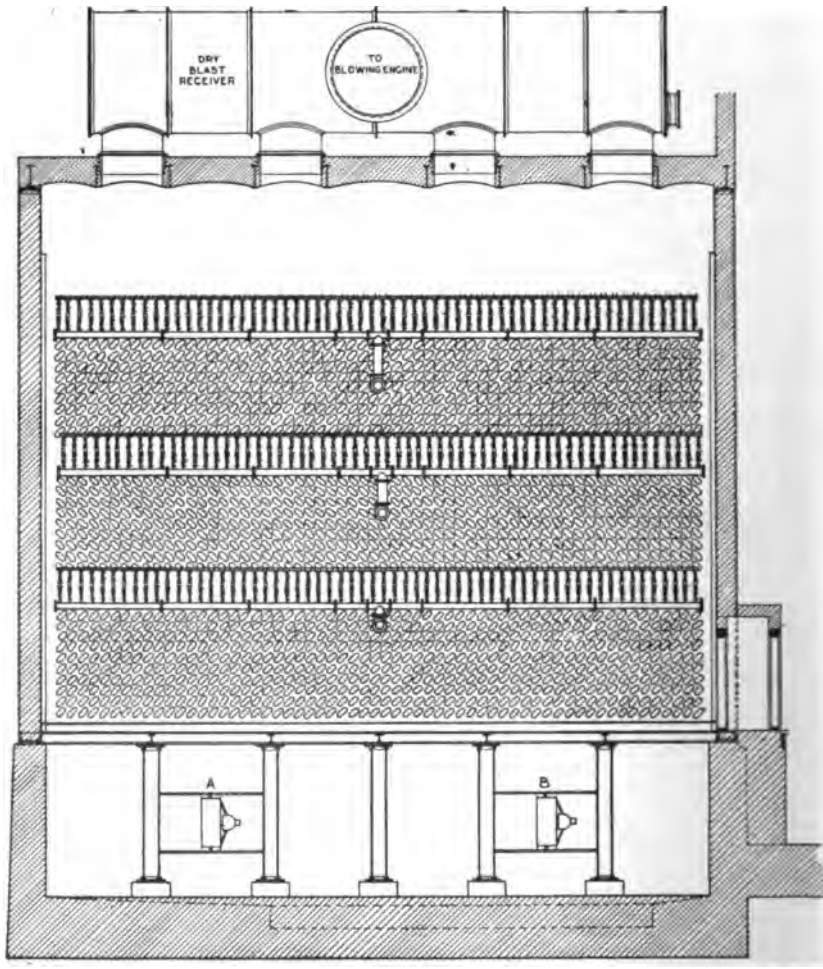
FIG. 2.—COMPRESSORS, CONDENSERS AND REFRIGERATING CHAMBER, SIDE ELEVATION.

transverse section. The refrigerating-chamber is lined on the inside with plates of compressed cork 2 in. thick.

The ammonia-machines are of the compressor type, and were built by the York Manufacturing Co., York, Pa. The dimensions are as follows: Diameter of the high-pressure steam-

cylinder, 28.5 in.; low-pressure, 56 in.; compressor cylinder, 22.5 in.; stroke, 36 in. Two compressors were installed in order to have one in reserve at all times, as a funace operating on uniformly dry air cannot be subjected to ordinary atmospheric

FIG. 3.

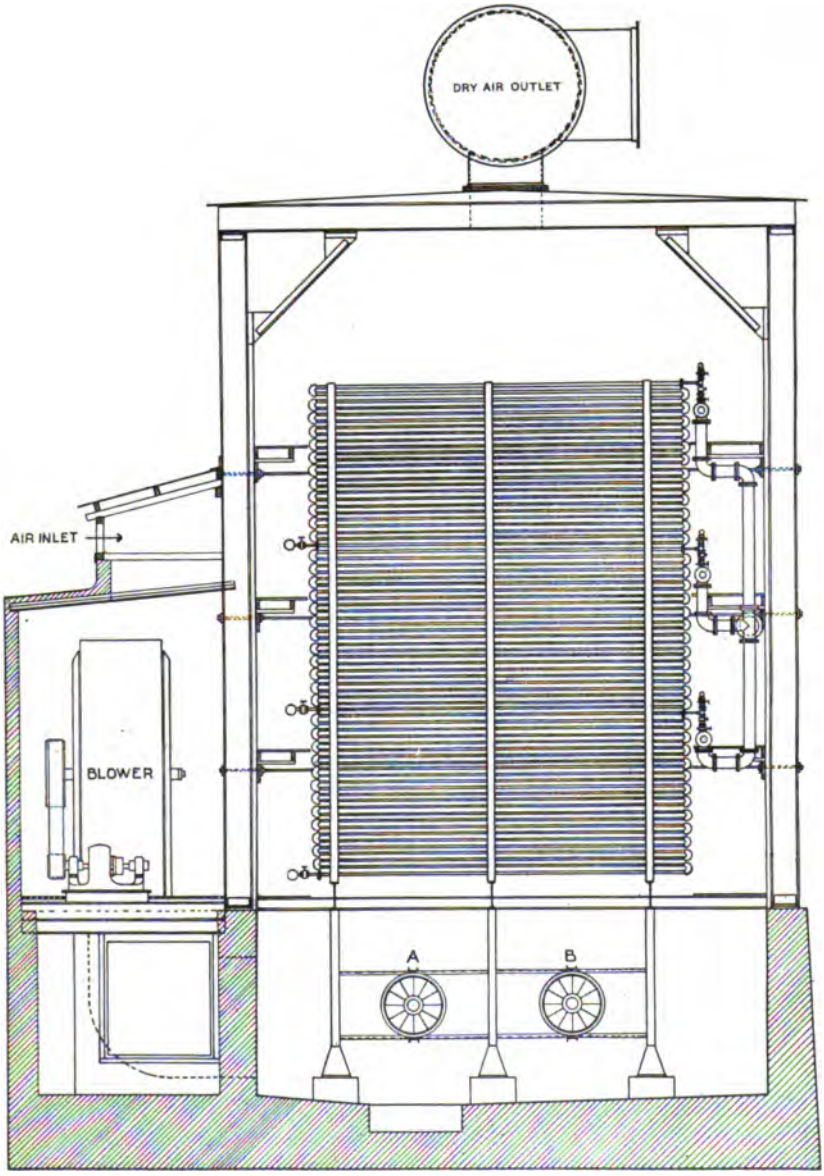


REFRIGERATING-CHAMBER, TRANSVERSE SECTION.

conditions without serious results, and frequently on very humid days the assistance of the second engine might be required. Each compressor has the capacity to melt 225 tons of ice.

Fig. 5 shows the brine-tank, containing 20 coils of pipe of

FIG. 4.



REFRIGERATING-CHAMBER, LONGITUDINAL SECTION.

the dimensions shown in the diagram. The coils are covered with calcium chloride brine having a sp. gr. of 1.21. The return brine from the refrigerating-chamber flows into the

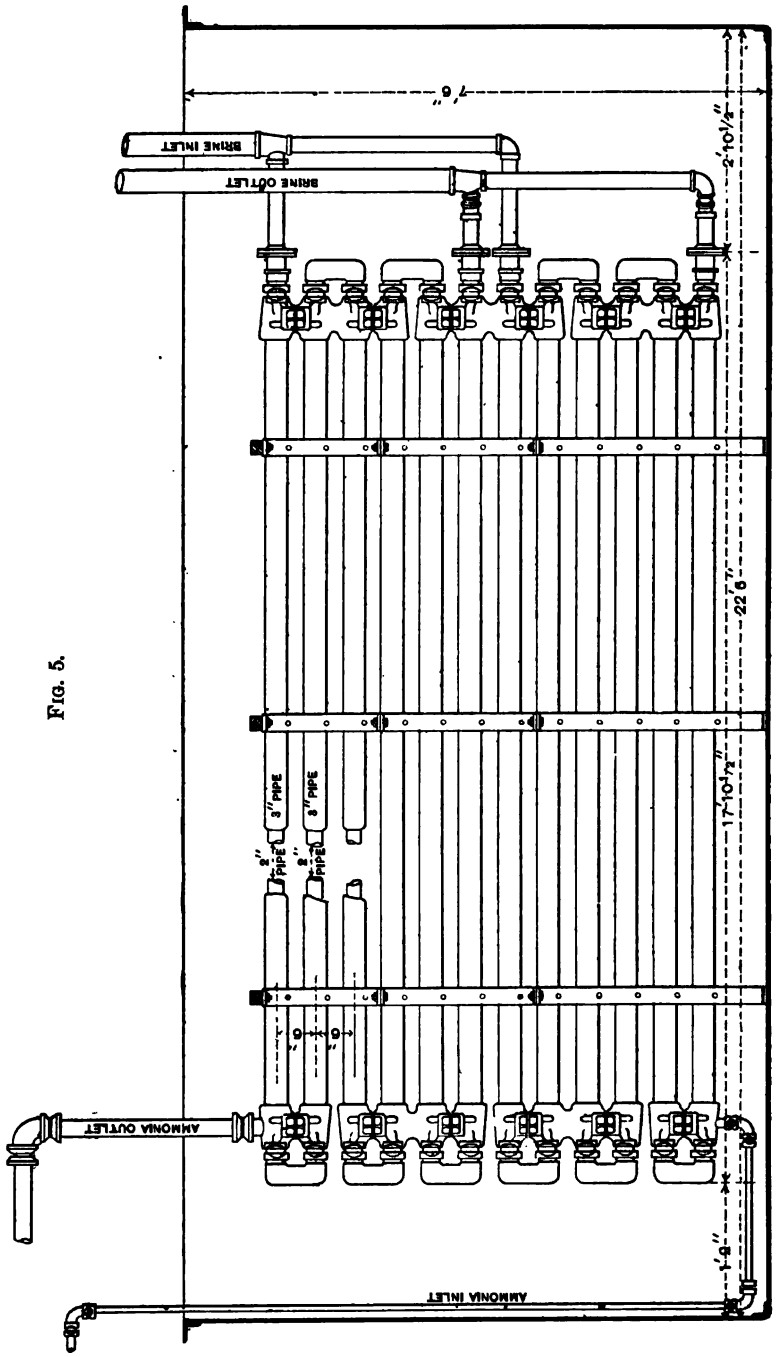


FIG. 5.

BRINE-TANK, DETAILS OF COILS OF PIPES.

top of the tank, is cooled by the ammonia expanding between the outer and inner pipes, withdrawn by a pump, and forced back through the pipe marked "brine-inlet" into the 2-in. or inner pipes (where it is cooled below the freezing-point), and thence into the coils in the refrigerating-chamber. The ammonia enters at the bottom of the pipes, thus traveling in the opposite direction from the brine, and by expanding between the 2-in. and 3-in. pipe, cools the brine, both in the tank and in the inner pipes; 40,000 gal. of brine are required in the system.

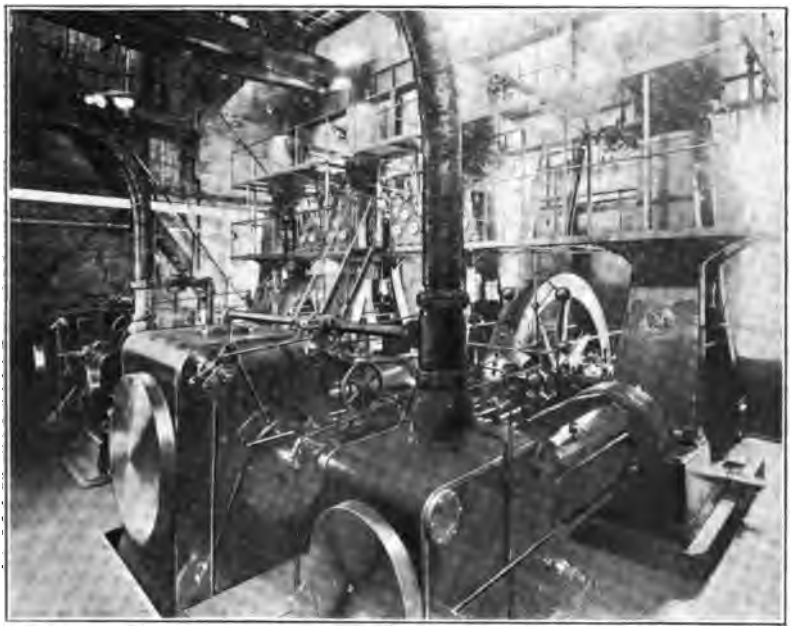
Figs. 3 and 4 show the arrangment of pipes in the refrigerating-chamber. There are, in each vertical line of coils, seventy-five 2-in. pipes 20 ft. long, and in the chamber sixty such vertical lines, the whole representing 90,000 linear ft. of 2-in. pipe in the chamber. The pipes in each vertical coil are placed in staggered position to insure better contact with the air. The series of coils is divided into three sections, each fed through a 4-in. header, and discharging into a 6-in. header and thence into a standpipe, from which the brine flows to the brine-tank, the feed being arranged so that the brine flows in a direction opposite to that of the air. As the space between the pipes would become gradually reduced through the accumulation of frost, which might diminish the efficiency of the blowing-engine, a blower was installed to force air into the refrigerating-chamber; and in order to secure a uniform distribution of air over the coils, revolving-fans (A and B, Figs. 3 and 4) were placed in the space underneath, so that all the coils would frost alike. The entering air, according to its humidity, deposits its moisture in the form of water or frost on the lower pipes and as frost only on the upper pipes, and passes from the top of the chamber to the blowing-engines at or below the temperature of freezing, and with a practically uniform content of moisture. When the pipes have become covered with frost the cold brine is shut off from several vertical lines of coil at once; by means of an auxiliary pump and line of pipe, brine that has been heated in a tank with steam is forced through; and in a few minutes the frost is melted. Connection is then made with the cold-brine system, and frost begins to deposit quickly. The frost which has been melted off the pipes collects in a trough in the basement floor, from which it flows into the supply-tank for the condenser.

The plant is constructed throughout in the most substantial manner, for it is obvious that an apparatus treating such an important element of the process as the atmosphere could not be practically applied to a modern furnace in an experimental way, but must of necessity be as ample in capacity and as substantial in construction as any of the accessories of the furnace-stack now in use.

In order to show the arrangement of different parts of the plant, four photographic illustrations are given. Fig. 6 shows the ammonia-compressors; Fig. 7, a view of the top of the brine-cooling tank with its pipe-connections; Fig 8, the fans under the refrigerating-coils, which are used to circulate the air; and Fig. 9, the frosted ends of the coils in the refrigerating-chamber.

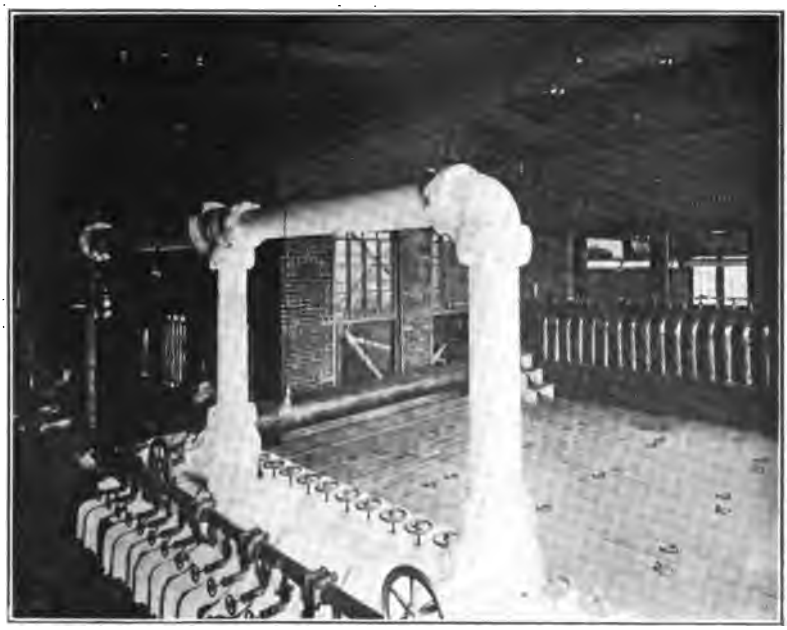
This dry-blast plant was put in operation August 11, 1904. The furnace was making a grade of iron suitable for the basic open-hearth furnace, containing less than 1 per cent. of silicon, with an ore-mixture consisting of 50 per cent. of Mesabi ore, the balance being soft hematites from Michigan. The mixture showed a yield by analysis of 53.5 per cent. of iron. The coke used was shipped from two mines and varied considerably in ash. The quantity of ash present in these two coals averaged 10.5 and 12.5 per cent. respectively. In order to obtain correct data from the use of the dry-blast, it was determined beforehand that no changes in any particular were to be made in the operation of the furnace, other than the introduction of dry air, and this has been rigidly adhered to. In the data following, a comparison is made between the operations of the furnace using dry air after August 11th and those from August 1st to 11th, when the furnace was using the atmosphere under ordinary conditions. A comparison with the previous month would show a greater economy in coke, but since a change was made in the ore-mixture in the latter part of July —which gave a lower coke-consumption per ton of iron—a comparison of data when using dry air with that obtained in August prior to its use, and with the same ore-mixture, would more accurately show the benefits derived. The burden on the furnace from August 1st to 11th inclusive was as follows:

FIG. 6.



AMMONIA COMPRESSORS.

FIG. 7.



TOP OF THE BRINE-COOLING TANK.

[To face p. 762]

FIG. 8.

REVOLVING ELECTRIC FANS UNDER REFRIGERATOR COILS.

FIG. 9.

FROSTED ENDS OF COILS IN THE REFRIGERATING-CHAMBER.

Coke,	10,200 pounds.
Ore,	20,000 pounds.
Stone,	5,000 pounds.

On August 11th a 5-per cent. increase in burden was put on the furnace and, later in the day, 33 per cent. of dry blast was used. As soon as this small quantity was introduced its effect was noticeable by a brightening of the tuyeres and an increasing temperature of the cinder. After this change in burden had come to work, the condition of the furnace being, if anything, still more satisfactory, an additional 5 per cent. of burden was put on, with confidence that an increased use of dry-blast would offset the increased duty on the furnace. From this period, on, the burden and volume of dry-blast were increased more slowly until, on August 25th, the furnace, using dry-blast entirely, had the following burden:

Coke,	10,200 pounds.
Ore,	24,000 pounds.
Stone,	6,000 pounds.

an increase in burden of 20 per cent. in two weeks. The record of the furnace from August 1st to 11th, prior to the use of the dry-blast, and from August 25th to September 9th inclusive, using all dry-blast, is shown in Table VII.

TABLE VII.—*Furnace Records, Without and With Dry Blast.*

WITHOUT DRY-BLAST.			WITH DRY-BLAST.		
Date.	Product.	Coke Consumption.	Date.	Product.	Coke Consumption.
	Tons.	Pounds.		Tons.	Pounds.
Aug. 1,	360	2,210	Aug. 25,	462	1,766
" 2,	367	2,112	" 26,	441	1,850
" 3,	372	2,084	" 27,	477	1,668
" 4,	373	2,133	" 28,	516	1,462
" 5,	386	2,008	" 29,	405	1,763
" 6,	340	2,280	" 30,	441	1,804
" 7,	347	2,116	" 31,	462	1,722
" 8,	360	2,012	Sept. 1,	472	1,729
" 9,	378	2,114	" 2,	472	1,642
" 10,	352	2,318	" 3,	458	1,648
" 11,	306	2,266	" 4,	421	1,841
			" 5,	450	1,813
Average,	358	2,147	" 6,	400	1,683
			" 7,	400	1,734
			" 8,	397	1,952
			" 9,	472	1,642
			Average,	447	1,726

Fig. 10 shows graphically the operations of each day, averaged with all preceding days, from Aug. 1 to Sept. 9, inclusive; the increase in output and reduction in coke-consumption corresponding to the increase in burden; the varying conditions of humidity from day to day, which represent the average humidity for each twelve-hour period; and the change in humidity after treatment in the dry-blast apparatus. While the reduction of moisture and its increased uniformity is considerable, it should not be lost sight of that this represents the beginning of operations, and there was still much to be learned with respect to the manipulation of the dry-blast plant.

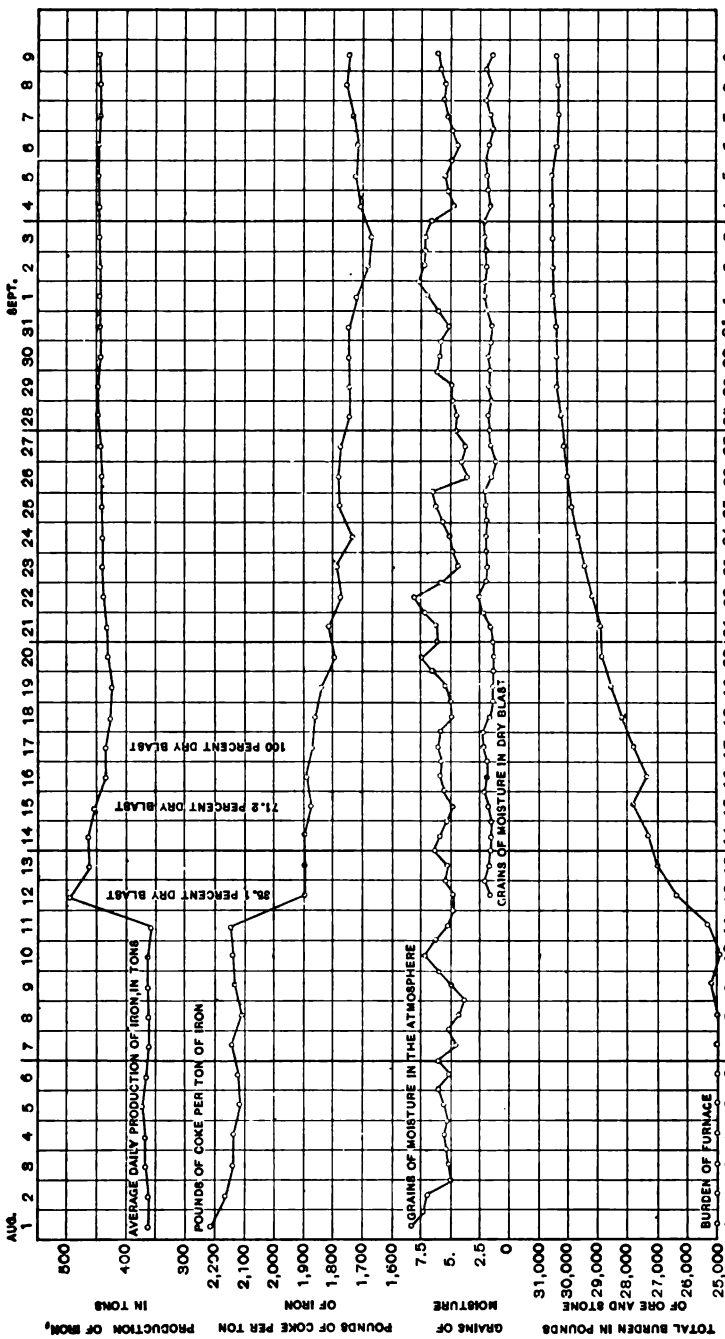
The effect of reducing and making more uniform the moisture in the blast was clearly shown when, during a period of excessive humidity extending over three days, a neighboring furnace charged during this period an extra quantity of coke and increased the quantity each day, in order to maintain the grade of iron, while the Isabella furnace, operating with dry-blast, was in no wise affected.

On September 10th, it was found necessary to make some repairs to the compressors and to make connections to a new brine-header for thawing-off the coils, and the burden was lightened accordingly. After these repairs had been made, the burden was again increased, and from September 17th to 30th, inclusive, the furnace showed an average daily output of 452 tons, with a coke-consumption of 1,729 lb. per ton of iron.

Of the changes made in the atmosphere by passing it through a refrigerating-chamber, the daily records of operations, set forth in Table VIII. (page 768), will give a very clear idea.

During thirteen days, the average moisture in the atmosphere was 5.66 grains per cu. ft.; and in the dry air 1.75 grains; 69 lb. of water were removed from the blast per ton of iron produced, which represents an average of 23,192 lb. (2,784 U. S. standard gallons) for the 24 hours. This weight was calculated from the volume of air blown into the furnace, as shown by piston-displacement. For four days during the above period the water, caught in the tank underneath the refrigerating-chamber, amounted to an average of 21,561 lb. (equivalent to 2,588 gal.) for the 24 hours, which is as close an agreement as could be expected, considering that the figures do not represent the same number of days, and that it is difficult to determine accurately the volume and humidity of the air sup-

FIG. 10.



CUMULATIVE RECORD OF FURNACE-OPERATIONS, AUGUST 1 TO SEPTEMBER 9, 1904. (Each day being averaged with all preceding days.)

TABLE VIII.—*Data of Refrigerating-Chamber Results.*

Time.	Temperature.		Grains of Water per Cu. Ft. Air.		Temperature.		Grains of Water per Cu. Ft. Air.		Temperature.		Grains of Water per Cu. Ft. Air.	
	Inlet.	Outlet.	Inlet.	Outlet.	Inlet.	Outlet.	Inlet.	Outlet.	Inlet.	Outlet.	Inlet.	Outlet.
6 A.M.	68	21	5.19	1.33	70	22	6.35	1.70	77	22	3.94	1.48
7 A.M.	68	20	5.02	1.24	71	22	6.78	1.77	.....	.....	4.08	1.29
8 A.M.	70	20	5.56	1.55	69	22	6.67	1.62	.....	.....	4.22	1.42
9 A.M.	73	20	5.37	1.46	73	22	6.78	1.70	71	25	4.85	1.36
10 A.M.	74	20	5.47	1.81	74	22	6.78	1.70	.....	.....	5.02	1.48
11 A.M.	77	20	5.56	1.53	77	23	6.67	1.70	.....	.....	5.19	1.55
12 M.	77	21	6.04	1.53	81	23	6.56	1.62	81	28	5.37	1.70
1 P.M.	80	21	6.04	1.42	78	24	6.56	1.70	.....	.....	4.85	1.62
2 P.M.	81	22	6.14	1.60	82	25	6.56	1.90	.....	.....	4.85	1.62
3 P.M.	81	23	5.74	1.60	81	24	6.19	1.74	84	29	5.02	1.70
4 P.M.	82	23	5.74	1.55	81	24	6.19	1.42	.....	.....	4.68	1.48
5 P.M.	82	22	6.04	1.62	80	24	6.14	1.48	.....	.....	4.85	1.60
6 P.M.	81	23	5.94	1.55	75	24	5.56	1.55	78	29	5.37	1.77
7 P.M.	80	23	5.74	1.62	72	24	5.94	1.70	.....	.....	5.37	1.62
8 P.M.	79	24	5.94	1.55	70	23	5.19	1.62	....	.....	5.56	1.70
9 P.M.	73	23	7.01	1.85	69	22	5.19	1.42	72	29	5.74	1.70
10 P.M.	73	22	6.78	1.70	68	21	5.19	1.55	.....	.....	5.74	1.77
11 P.M.	73	23	6.78	1.70	66	20	3.94	1.77	.....	.....	5.74	1.62
12 NGT.	73	23	7.01	1.70	62	20	3.54	1.62	66	28	5.56	1.70
1 A.M.	73	23	6.78	1.70	59	18	3.41	1.42	.....	.....	4.85	1.70
2 A.M.	74	23	7.01	1.70	57	17	3.54	1.13	.....	.....	5.37	1.70
3 A.M.	73	23	6.78	1.70	56	16	3.18	1.13	64	27	5.19	1.48
4 A.M.	73	23	6.78	1.48	56	16	3.18	0.99	.....	.....	5.19	1.36
5 A.M.	73	23	6.78	1.48	53	14	2.85	1.06	.....	.....	4.85	1.48

plied in a given period. It is found sufficient in practice to thaw the frost off the pipes every three days. The coils are divided, for the purpose of thawing-off, into three sections, each representing the same number of coils, and a section is thawed each day; and in this way the work of refrigeration is not interfered with.

When the dry-blast was supplied to the furnace, it became necessary to reduce the revolutions of the blowing-engines, since the air supplied to the engines was lower in temperature than the natural atmosphere and contained more oxygen per cu. ft., and the tendency of the furnace was to drive too fast. Before applying the dry-blast the engines were running at 114 revolutions and supplying 40,000 cu. ft. of air per minute; the revolutions were gradually reduced to 96, thereby reducing the volume of blast over 6,000 cu. ft. per minute and increasing the efficiency of the engines by 14 per cent. With dried blast, 96 rev. per min. of the blowing-engines burned nearly 1 per cent. more coke and produced 89 tons more pig-iron in 24 hours

than 114 rev. per min. with atmospheric air. The reduction in the revolutions resulted in a gain of 150 degrees in the temperature of the blast, which, even with this increase, through lack of area in the waste-gas ports of the stove, did not average above 870 degrees.

The average analysis of the gas for ten days prior to the introduction of the dry-blast showed:—CO, 22.3 per cent.; CO_2, 13 per cent., with an average temperature of 538 degrees. Later, with dry-blast used entirely, the average analysis was:—CO, 19.9 per cent; CO_2, 16 per cent., with an average temperature of 376 degrees. This reduction in temperature of 162 degrees is a necessary consequence of the greater concentration of heat in the hearth by the dry-blast combustion and the greater weight of burden heated by the gas, and represents an important saving of heat in the furnace.

The dry blast has resulted in economies in several other directions. In the use of Mesabi ore, which is very fine in structure, the waste of ore-dust carried by the escaping gases is quite large, and at many furnaces it has become quite burdensome. At the Isabella furnace, before dry-blast was used, it amounted to 5 per cent. of the ore charged; and it has been reduced, through the greater uniformity in the furnace-working, effected by the dry blast, to less than 1 per cent.

The saving in coke-consumption reduces the phosphorus in the metal; and this, in making Bessemer iron, permits the use of ores higher in phosphorus. As the Isabella furnace was making basic iron, it was an advantage to keep the silicon as low as possible, provided the sulphur was kept low; and the absence of irregularities in the furnace-operations resulting from the dry blast permitted the keeping of the silicon at a lower range without increasing the sulphur. It has been generally observed by furnace-managers that when the silicon is lowered through increased humidity in the atmosphere, a leaking tuyere, or through other causes, the sulphur is rapidly increased; but it has been found in using the dry blast, that when the hearth-temperature was suddenly lowered, principally from accretions on the bosh reaching the hearth, the sulphur did not increase; and in this respect the furnace has shown a remarkable uniformity in composition of the metal produced.

Mention has been made of the saving effected at the blowing-engines through a reduction in the number of the revolu-

tions, which has an important bearing on the expenditure for power in operating the machines in the dry-blast plant. Prior to the use of the dry-blast plant, the average developed by each engine was 900 i.h.p. From the cards taken when the furnace was supplied with dry-blast, this average was 671 i.h.p., a difference of 229 i.h.p. per engine, or 687 i.h.p. for the three engines. Cards were also taken from the ammonia-compressors, the compression and back-pressure being kept as nearly as possible to the best working-condition. Running at 45 revolutions, which would probably represent the average for the year, each engine developed 230 i.h.p., or 460 i.h.p. for the two engines; the fans, together with the brine- and water-pumps, are well covered by allowing for them 75 i.h.p., making a total of 535 i.h.p. Comparing this with the power saved in the blowing-engine room, there appears a net saving, after allowing for the operation of the dry-blast plant. These figures, however, may not represent accurately the difference in power-consumption, as the blowing-engines were indicated at different times and the first test was taken with a blast-pressure on the furnace of 17 lb., while the test made with the dry blast was 15 lb., hence the figures given above might require some modification, as the effect of dry blast on blast-pressure is not yet fully determined. The increase of uniformity in the working of the furnace, obtained through the dry blast, would result in a decrease of the blast-pressure; and I think, in any event, the saving in power-consumption in the blowing-engine room would nearly or quite compensate for the requirements of the dry-blast plant.

The application of the dry blast to the blast-furnace has shown, in addition to the economies effected, that the furnace can be controlled with precision; that it works with greater regularity; and that, in consequence, the product is uniform with respect to grade and composition, which makes the dry blast particularly valuable in the making of foundry-iron, which is marketed by grade. An increase or decrease in blast temperature has a definite effect, and can be relied on to accomplish the desired result.

The dry-blast plant here described has been in regular operation since August 11, 1904. It started without a hitch and no difficulties have developed in any direction. Some modification in construction has been indicated as the result of the op-

eration of the plant which would further reduce the moisture and add to its uniformity, but so far the changes suggested have been slight.

As above shown, the application of the dry blast to the blast-furnace has effected various economies and produced a more uniform metal. Probably its further application to the Bessemer converter would result in great benefit, since, in that apparatus, air is used in large quantities and the varying humidity affects the temperature of the charge and in consequence the quality of the steel. The metal from the metal-mixer employed in many Bessemer works is remarkably uniform; and the additional uniformity secured through the use of dry air would be of further advantage. In our American practice, a higher silicon is required in the summer months to maintain the temperature of the blow, in which period it is also more expensive to maintain the right amount of silicon in the pig-iron. With the use of the dry blast in the converter the proper temperature could be secured with a lower silicon in the metal, and this in turn would further reduce the coke-consumption at the furnaces. In other processes where air is used in large quantities—particularly in copper- and lead-smelters and copper-converters, in the open-hearth furnace and in cupolas—it seems probable that the use of dry-air would effect important economies, and its application to gas-producers, by-product coke- and charcoal-ovens, for the extraction of the moisture, would be very beneficial.

In the development of this process, which has occupied many years, during part of which time I was not directly in charge of blast-furnaces, much of the work has been necessarily entrusted to others. I desire to express my indebtedness to Mr. James Scott, General Superintendent of the Lucy and Isabella furnace-plants, and Mr. John P. Collins, Superintendent of the Isabella, for their valuable assistance in the application of the process; to Mr. Bruce Walter, Engineer at the works named, who conducted with great skill the later experiments, and supervised the construction of the dry-blast plant at the Isabella furnaces; and to Mr. John C. Greenway, then in the engineering department of the Duquesne steel-works, who conducted many of the earlier experiments.

The Influence of Carbon, Phosphorus, Manganese and Sulphur on the Tensile Strength of Open-Hearth Steel.

BY H. H. CAMPBELL, STEELTON, PA.

(Lake Superior Meeting, September, 1904.)*

MANY attempts have been made to write a formula by which to calculate the strength of steel from its chemical composition, but most of these endeavors have failed because there were too many disturbing conditions. It is idle to collect from the pages of trade-papers, books, or the proceedings of scientific societies, a multitude of observations. The combination of such results will simply show that steel of the same composition will vary in tensile strength through wide limits—a fact that has been known for generations. That cold-working, overheating, and many another form of heat-treatment alter fundamentally the strength of steel, is ancient history, although it is only recently that the microscope has pointed out the road to an explanation of the phenomena.

It may be urged that the microscopic structure must be taken into consideration in any formula giving the ultimate strength, but from the standpoint of the present investigation this is unnecessary. We are trying to determine primarily, not what changes in ultimate strength may be made by variations in the condition of carbon, but what effect is produced by changes in the amount of carbon, when its condition remains constant. For such an inquiry, in order that the condition of the carbon should remain as nearly uniform as possible, it is essential that all test-pieces be made under the same conditions; and it is believed that the tests described in this paper satisfy that requirement. The investigations were made at the works of the Pennsylvania Steel Company, Steelton, Pa.; the ingots from which the tests were made were 6 in. square in every case; they were heated in the same furnace and forged at the same ham-

* Read by title at the Lake Superior meeting, but first presented at the New York meeting of the Iron and Steel Institute (October, 1904), and here published under a mutual agreement between the Councils of the two Institutes.

mer into billets of the same size; these billets were reheated in the same furnace by the same men and rolled in the same set of rolls into 2- by $\frac{3}{8}$-in. bars of about the same length. These were cooled under the same conditions, broken in the same machine by the same men, and analyzed in the same laboratory by the same staff.

First Investigation.

About ten years ago extensive calculations were made on such bars at Steelton by the method of least squares.[1] In this case the bars of similar composition were grouped together, and the carbon of the group was determined by combustion upon a sample containing an equal amount of drillings from each bar. Such a grouping of the tests would give rise to error if unlike bars were put together; so that this was avoided as far as possible. A greater cause of trouble lies in the fact that, in the method of least squares, any error in one factor affects the other factors. If for any reason the value of phosphorus comes out higher than the real truth, the value of carbon may suffer accordingly. Still greater is the objection that no factor can be used in the work unless it really has a decided influence on the results. This was illustrated when sulphur and copper were used as factors, the values found for them being absurd.

In acid steels, it appeared that carbon and phosphorus were the important elements to be considered; and it was assumed that the strength of steel was made up of the effect of a certain content of iron, plus the effect of a certain content of carbon, plus the effect of a certain content of phosphorus. Thus in a metal containing 0.20 per cent. of carbon and 0.08 per cent. of phosphorus, and having a tensile strength of 70,000 lb. per sq. in., we may write the following equation (the iron being determined by difference):

$20\ C + 8\ P + 9{,}972\ Fe = 70{,}000$; C, P and Fe being constants, representing the effect upon the tensile strength in pounds per square inch of 0.01 per cent. of carbon, phosphorus and iron respectively. In this way each group of tests furnished a formula and the combination of these by the method of least squares gave the following results:

[1] Full details of the work will be found in *The Manufacture and Properties of Iron and Steel*, by H. H. Campbell, 2d Ed., pp. 482–528. Engineering & Mining Journal Company, New York, 1904.

For Acid Steel: (C = 1,210; P = 890; Fe = 38,600.)

38,600 Iron + 1,210 Carbon + 890 Phosphorus + R = Ultimate Strength.

For Basic Steel: (C = 950; P = 1,050; Fe = 37,430, and Mn = 85.)

37,430 + 950 Carbon + 1,050 Phosphorus + 85 Manganese + R = Ultimate Strength.

The iron, carbon, phosphorus and manganese are expressed in units of 0.01 per cent., and R, as well as the ultimate strength, in pounds per square inch. R is a variable, to allow for heat-treatment. In angles and plates, about $\frac{3}{8}$-in. or $\frac{1}{2}$-in. in thickness, and finished at a fairly high temperature, R is zero.

In this formula, Fe, 38,600, represents the value independently determined for pure iron. From a mathematical standpoint there can be no objection to including iron as one of the factors in the problem, but practically there are the following reasons to the contrary:

1. There is a doubt whether the real basis of strength varies with each increase or decrease in the total metallic iron. It may be that the datum-plane is the same, whether the steel contains 99.6 per cent. or 99.1 per cent. of iron.

2. Since the iron is determined by difference, all the errors in determining carbon, manganese and phosphorus, as well as the total contents of sulphur, copper, silicon, and other elements, may make a composite error of no small moment; and this is all embodied in the figure representing iron.

3. The range of variation in the percentage of iron is not sufficiently great to give a good working-basis.

Notwithstanding all these objections to the methods of determination, the derived formulæ given above have been of the utmost practical importance. They have been applied to every heat of steel made by the Pennsylvania Steel Company for the last ten years; and there has rarely been, in the ultimate strength of soft steels, a difference of more than 2,500 lb. per sq. in. between the result given by the formula and the record of the testing-machine. In most cases the error has been much less than 1,500 lb.; and so great is our confidence in the formulæ, that chemical determinations are always repeated when they are not confirmed by the machine-test. In view of such an experience in every-day commercial work, it would be

rash to say that the method is entirely wrong, or that the formulæ do not represent actual conditions.

Second Investigation.

To check the first investigation, two entirely new series of bars were collected: One of nearly seven hundred from acid heats, and the other of eleven hundred from basic heats. Duplicate determinations were made on each bar for phosphorus and manganese. The carbon was determined in three ways: (1) the bar was analyzed by combustion (duplicate tests being made in case of doubt); (2) the bar was analyzed by the color-test; (3) a piece of the ingot from which the bar had been made was cut off at the hammer, and analyzed by the color-test.

Three bars were pulled on each heat, two on one testing-machine and one on another. The figure used is the average of the results obtained on the two machines, not the average of the three bars.

In order to plot the data, all heats were combined which showed carbon from 0.075 to 0.125 per cent.; from 0.125 to 0.175 per cent., and so on—making a division for each additional 0.05 per cent. of carbon. Table I., on the following page, gives the list of groups thus formed.

The division of the groups according to the way in which the carbon was determined is important, because many heats may change their grouping according to the way the carbon is determined; thus if a heat showed 0.12 per cent. of carbon by combustion, it would appear in Fig. 1 in line AA, at the point representing the range between 0.075 and 0.125 per cent. of carbon; while, if the color-determination showed 0.14 per cent., it would appear in the line BB at the point representing the range between 0.125 and 0.175 per cent. In this way the three sets of lines may be viewed as the result of three independent investigations.

The lines AA, BB and CC in Fig. 1 are not plotted directly from Table I., but the data have been combined by recognized scientific methods to allow for the unequal number of heats in the groups. Thus by combining groups 1, 2 and 3, we get the first point of AA; from groups 2, 3 and 4 we get the second point; and so on. The result of this combination gives Table II. The lines AA, BB, etc., plotted therefrom, are more nearly

TABLE I.—*List of Groups Used in Determining the Effect of Carbon, Phosphorus and Manganese.*

Division.	Number of Heats.	Average Chemical Composition.			Ultimate Strength.
		Carbon.	Phosphorus.	Manganese.	
		Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. In.
A, acid test-bars; carbon by combustion.	50	0.1118	0.0545	0.408	58,012
	131	0.1463	0.0567	0.437	61,039
	58	0.1995	0.0579	0.475	66,809
	22	0.2463	0.0563	0.484	70,736
	50	0.3065	0.0476	0.528	79,058
	120	0.3501	0.0466	0.537	83,093
	103	0.4000	0.0400	0.518	87,156
	86	0.4491	0.0376	0.520	92,824
	42	0.4961	0.0363	0.519	98,224
	8	0.5460	0.0354	0.495	102,346
	6	0.5863	0.0330	0.493	107,398
B, acid test-bars; carbon by color.	45	0.113	0.0545	.....	58,535
	164	0.145	0.0568	.....	62,407
	47	0.197	0.0560	.....	67,052
	8	0.249	0.0527	.....	72,728
	36	0.304	0.0494	.....	80,776
	53	0.352	0.0380	.....	86,369
	45	0.395	0.0358	.....	92,759
	18	0.444	0.0330	.....	98,576
	2	0.480	0.0360	.....	103,120
C, acid test-ingots; carbon by color.	34	0.118	.....	.....	57,599
	160	0.145	.....	.....	61,189
	61	0.190	.....	.....	67,482
	17	0.250	.....	.....	74,239
	84	0.307	.....	.....	80,491
	160	0.346	.....	.....	85,073
	98	0.397	.....	.....	91,434
	59	0.446	.....	.....	97,439
	15	0.507	.....	.....	105,656
D, basic test-bars; carbon by combustion.	135	0.0451	0.0082	0.243	46,703
	125	0.0974	0.0084	0.422	50,013
	134	0.1521	0.0116	0.436	55,650
	246	0.2044	0.0113	0.472	61,236
	263	0.2484	0.0110	0.474	64,744
	125	0.2935	0.0106	0.464	68,307
	27	0.3413	0.0113	0.461	72,065
	11	0.3932	0.0120	0.499	78,625
	1	0.4310	0.0070	0.390	83,305
E, basic test-bars; carbon by color.	173	0.047	0.0076	.....	47,084
	96	0.093	0.0100	.....	51,228
	189	0.154	0.0122	.....	58,202
	322	0.200	0.0118	.....	63,184
	235	0.248	0.0116	.....	65,813
	51	0.288	0.0125	.....	70,786
	3	0.343	0.0087	.....	79,252
F, basic test-ingots; carbon by color.	131	0.057	.....	.....	46,431
	131	0.093	.....	.....	49,917
	152	0.150	.....	.....	56,264
	365	0.203	.....	.....	62,241
	210	0.246	.....	.....	66,401
	72	0.295	.....	.....	71,011
	10	0.350	.....	.....	80,013
	2	0.400	.....	.....	80,272

TABLE II.—*Combination of Data in Table I, by Groups of Three, to Obtain Construction Points for the Lines in Fig.* 1.

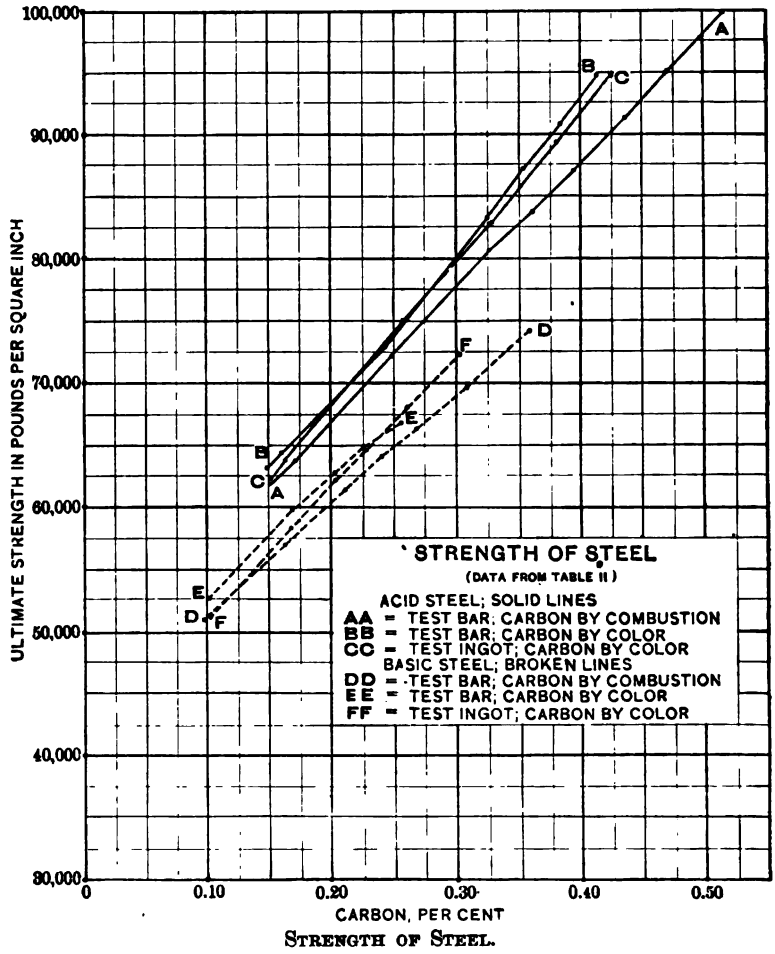
Class.	Chemical Composition.			Ultimate Strength.
	Carbon.	Phosphorus.	Manganese.	
	Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. In.
A, acid test-bars; carbon by combustion; line AA.	0.1520	0.0565	0.440	61,806
	0.1713	0.0570	0.453	63,637
	0.2486	0.0537	0.497	72,185
	0.3268	0.0480	0.529	80,626
	0.3609	0.0443	0.528	83,886
	0.3943	0.0419	0.526	87,155
	0.4357	0.0384	0.519	91,278
	0.4693	0.0371	0.518	95,052
	0.5130	0.0358	0.513	99,795
B, acid-test bars; carbon by color; line BB.	0.1489	0.0562	0.443	63,072
	0.1600	0.0564	0.453	64,379
	0.2437	0.0541	0.491	72,980
	0.3255	0.0434	0.519	83,168
	0.3534	0.0403	0.515	87,012
	0.3827	0.0364	0.513	90,742
	0.4112	0.0351	0.506	94,689
C, acid test-ingots; carbon by color; line CC.	0.152	.....	.....	62,216
	0.164	.....	.....	63,734
	0.257	.....	.....	74,937
	0.327	.....	.....	82,893
	0.351	.....	.....	85,771
	0.381	.....	.....	89,341
	0.423	.....	.....	94,734
D, basic test-bars; carbon by combustion; line DD.	0.0978	0.0094	0.366	50,834
	0.1639	0.0107	0.450	57,001
	0.2115	0.0113	0.465	61,502
	0.2403	0.0110	0.471	64,086
	0.2681	0.0109	0.470	66,297
	0.3081	0.0108	0.466	69,626
	0.3582	0.0113	0.469	74,203
E, basic test-bars; carbon by color; line EE.	0.1010	0.0101	0.384	52,540
	0.1688	0.0116	0.458	59,739
	0.2036	0.0118	0.466	62,750
	0.2260	0.0118	0.468	64,839
	0.2564	0.0117	0.469	66,830
F, basic test-ingots; carbon by color; line FF.	0.102	.....	.....	51,150
	0.168	.....	.....	58,339
	0.204	.....	.....	62,195
	0.227	.....	.....	64,575
	0.262	.....	.....	68,010
	0.304	.....	.....	72,303

representative of the true conditions, and are straighter than lines would be if plotted directly from Table I.

In Fig. 1 are shown all six lines. The line BB, founded on the color-determination of carbon in the acid test-bar, agrees very closely with CC, founded on color-carbons from the acid test-ingot. In the same way, but less closely, the line EE, representing color-carbons on the basic bar, agrees with the line

FF, representing color-carbons on the basic ingot. This additional investigation of the test-ingot was undertaken simply as a check on the other results; and, since it agrees as nearly as could be expected with the color-records on the bar, no further mention will be made of its results, comparison being made

FIG. 1.



STRENGTH OF STEEL.

only between the combustion- and the color-work on the test-bar. It may be well, however, to note that the line FF does not dip down at the upper end like EE, and this fact, taken in conjunction with the straightness of all the other lines, indicates that this dip in EE must be caused by errors in determination.

In the acid steels there is quite a difference between AA

and BB, and in the basic steels between DD and EE. For a given amount of carbon, the results on the bar when analyzed by the color-method show a higher tensile strength than when carbon is determined by combustion, because a certain amount of temper-carbon is found by combustion which has little effect upon the ultimate strength. Determinations by color are often unsatisfactory; but when all the work is done in one laboratory, the results may be looked upon as comparable, and the reasonable straightness of the lines, as plotted, is good evidence that the chemical work was reliable.

The lines AA, BB, etc., take no account of variation in the content of phosphorus or of manganese. It is well known that phosphorus in small proportions adds to the tensile strength; in the first investigation by the method of least squares it was found that 0.01 per cent. raised the strength of acid steel 890 lb., and basic steel, 1,050 lb. per sq. in.; and experience has indicated that these values are very near the truth. In the present investigation the value of carbon is first determined, and then that of manganese and phosphorus, but in order to find the value of carbon accurately it is essential to know the influence of both manganese and phosphorus. This makes necessary the method of successive approximations, the values found in the first approximation being used in the second, and so on until the changes made in values are unimportant. In the present case the methods used avoid to some extent the dependence of one determination upon another. Thus in the line AA, carbon is the one great variable; the proportions of phosphorus and manganese are not constant, but the groups of high-carbon steel contain about the same amount of manganese and phosphorus as the groups of low-carbon steel, and hence the line will give a provisional value of carbon. The general trend is determined by stretching a thread along its length and noting the tangent made with the horizontal. In this way the line AA indicates a value for carbon of about 1,050 lb. for each 0.01 per cent.; allowances have yet to be made for the effect of phosphorus and manganese, but this figure serves as a working basis for similar provisional estimations of the other elements.

In explaining the method used to determine the value of phosphorus and manganese, no mention will be made of these

provisional values, the figures given being in each case the final results.

The Effect of Phosphorus on Acid Steel.

The investigation into the effect of phosphorus will be confined to acid steel, for in the basic steels under consideration the proportions of phosphorus were so low that the differences were almost within the limits of chemical error. Two methods have been used, one serving as a check upon the other.

First Method.—Referring to Table I, under the head of acid steels, carbon determined by combustion, the first group is composed of 50 heats averaging 0.1118 per cent. of carbon, 0.0545 per cent. of phosphorus, 0.408 per cent. of manganese and 58,012 lb. per sq. in. in ultimate strength. These 50 heats were separated into two groups, of high- and low-phosphorus respectively. The terms "high" and "low" are relative, and signify that the heats were arranged in a column according to their phosphorus-content, the upper half of the list being called "high" and the lower half "low." The result of this process applied to each group gave in each case two equal divisions showing certain differences in carbon, in manganese, in phosphorus and in ultimate strength. If the carbon and manganese had been uniform, the effect of phosphorus could have been found by simple division; but since they both varied, it was necessary to make allowance for them. If the difference in carbon were of any great amount a considerable error might be introduced; but as each group before division included only such heats as were within a range of 0.05 per cent. of carbon, the difference in this element in the high-phosphorus and low-phosphorus heats is in each case in the third decimal place.

Table III shows the first method of finding the value of phosphorus from the acid-steel bars in Table I. In each group we find the difference in ultimate strength, then allow for the variation in carbon, and in manganese, and find the difference in ultimate strength due to phosphorus alone; then divide by the variation in phosphorus and obtain the effect of one unit of phosphorus in the one group under consideration. If all the groups were composed of the same number of heats, it would suffice to take an average of these values of phosphorus, but as each group is made up of a different number, it is necessary to

make a true average by multiplying the value found for each group by the number of heats in the group, and divide the sum of these results by the total number of heats. In this way the first method gives a value for phosphorus in acid steel of 807 lb. for 0.01 per cent.

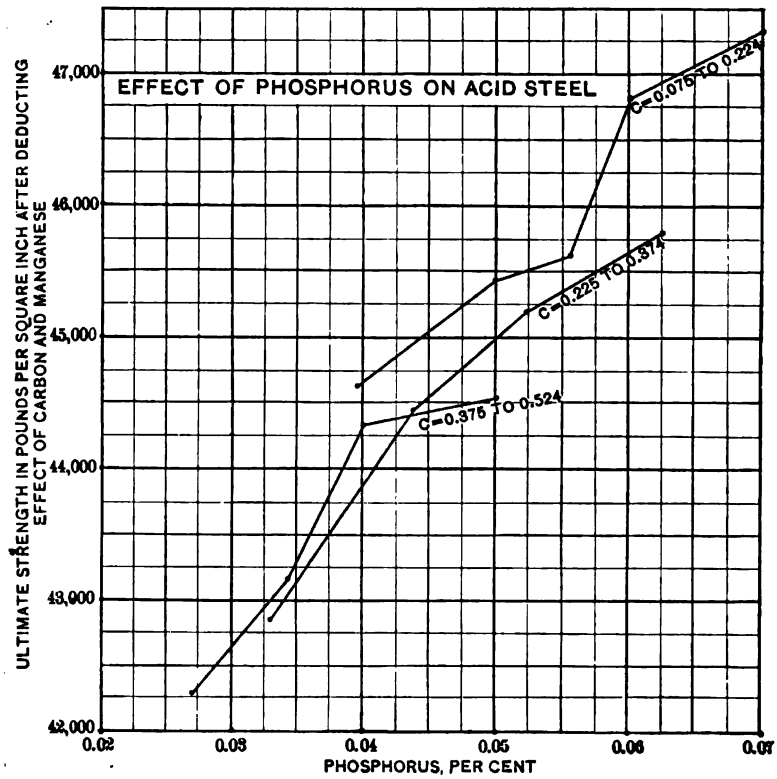
TABLE III.—*Division of Heats to Determine the Effect of Phosphorus upon Acid Steel.*

NOTE.—In the sixth column an allowance is made of 1,000 lb. for 0.01 per cent. of carbon, and for manganese according to the sliding scale in Table VII.

Relative Phosphorus.	Chemical Composition.			Ultimate Strength.			Number of Heats in Group.	Product of Last Two Columns.
	Carbon.	Phosphorus.	Manganese.	Actual Records.	Difference Due to Phosphorus Alone.	Effect of 0.01 Per Cent. of Phosphorus.		
	Per Cent.	Per Cent.	Per Cent.	Lb. per Sq. Inch.	Lb. per Sq. Inch.	Lb. per Sq. Inch.		
High	0.1121	0.0635	0.405	58,543				
Low	0.1114	0.0456	0.411	57,481				
Diff	0.0007	0.0179	0.006	1,062	1,045	584	50	29,200
High	0.1470	0.0652	0.444	62,088				
Low	0.1457	0.0483	0.432	60,006				
Diff	0.0013	0.0169	0.012	2,082	1,808	1,070	131	140,170
High	0.1963	0.0667	0.481	67,443				
Low	0.2026	0.0490	0.470	66,174				
Diff	0.0063	0.0177	0.011	1,269	1,723	973	58	56,434
High	0.2459	0.0658	0.489	71,292				
Low	0.2467	0.0484	0.481	70,275				
Diff	0.0008	0.0174	0.008	1,017	937	539	22	11,858
High	0.3065	0.0564	0.546	80,111				
Low	0.3065	0.0399	0.509	78,005				
Diff		0.0165	0.037	2,106	1,218	738	50	36,900
High	0.3466	0.0545	0.540	83,669				
Low	0.3543	0.0372	0.534	82,410				
Diff	0.0077	0.0173	0.006	1,259	1,861	1,076	120	129,120
High	0.3969	0.0470	0.520	87,104				
Low	0.4031	0.0328	0.516	87,209				
Diff	0.0062	0.0142	0.004	−105	387	273	103	28,119
High	0.4511	0.0441	0.524	93,673				
Low	0.4473	0.0311	0.517	91,975				
Diff	0.0038	0.0130	0.007	1,698	1,066	820	86	70,520
High	0.4949	0.0407	0.521	98,318				
Low	0.4970	0.0327	0.517	98,145				
Diff	0.0021	0.0080	0.004	173	226	282	42	11,844
High	0.5430	0.0433	0.508	104,047				
Low	0.5508	0.0275	0.483	100,645				
Diff	0.0078	0.0158	0.025	3,402	3,082	1,951	8	15,608
High	0.5815	0.0415	0.500	109,367				
Low	0.5898	0.0288	0.490	106,412				
Diff	0.0083	0.0127	0.010	2,955	3,321	2,615	6	15,690
Total							676	545,463
Average								807

Second Method.—The bars were classified as before according to carbon, and then each of these main groups was subdivided according to phosphorus. Heats containing 0.03 per cent. of phosphorus constituted one group; those with 0.031 per cent. another; those with 0.032 per cent. another, and so on. Having made a list of these groups they were put together so as to give four or five points with about an equal number of heats in

FIG. 2.



EFFECT OF PHOSPHORUS ON ACID STEEL.

each, the result being shown in Table IV. In the last column is given what may be called the base, or the strength of the iron and phosphorus after allowing for carbon and manganese; this last column is plotted in Fig. 2. By combining the groups so as to rectify the lines by the method used in Table II, it will be found that in the line representing heats ranging between 0.075 and 0.224 per cent. of carbon, the phosphorus has a

value of about 860 lb. for each 0.01 per cent.; in the range from 0.225 to 0.374 per cent. of carbon, the value is 940 lb.; between 0.375 and 0.524 per cent. of carbon, it is 1,290 pounds. This would indicate that as the percentage of carbon increases, the effect of each unit of phosphorus increases, but the difference is so unimportant and the margin of certainty so narrow that it will be better to make a true average of the three values. There were 239 heats giving a value of 860 lb., 192 heats giving 940 lb., and 231 heats giving 1,290 lb., so that the true average is 1,033 pounds. For the sake of simplicity the value of 0.01 per cent. of phosphorus will be taken as 1,000 pounds.

In reducing to a zero-base, as in the last column of Table IV, there will be certain errors, since the values of carbon and manganese are not inerrant; but the original classification into groups of about the same carbon minimizes the disturbing effect. Thus in Table IV the first main division has five units; the highest carbon is 0.1540 per cent. and the lowest 0.1491 per cent., a variation of 0.0049 per cent. Carbon has been valued at 1,000 lb. for 0.01 per cent., and if perchance that value is in error by 50 lb. the results determined from that division of the table will be wrong by only 50 x 0.49 = 25 lb. The last column shows a strength of 47,328 lb. for one base and 44,616 lb. for the other, a difference of 2,712 lb., so that the assumed error of 50 lb. in the value of carbon produces an error of only 1 per cent. in the value of phosphorus in this particular division. This argument applies also to the determination of the other elements in both acid and basic steel.

Another important consideration applying equally to the work on phosphorus and on manganese is the concordance of results obtained from different divisions. A general average obtained by grouping any data into two primal divisions gives conclusions of very limited value, but in this paper the practice is followed of subdividing in order to compare results. Thus from three independent lines of Fig. 2 the values of phosphorus varied from 860 to 1,290. It is quite possible that these variations were not accidental and that the variation represents a law of increasing effect with higher carbons, but leaving all this aside, it is certain that three separate determinations roughly agreeing with one another establish with reasonable certainty the general fact that 0.01 per cent. of phos-

phorus has a strengthening effect of somewhere about 1,000 lb. The validity of the conclusions is immeasurably superior to one based on a general average.

TABLE IV.—*Classification of Acid Heats According to Content of Phosphorus.*

NOTE.—In the last column a value of 1,000 lb. is given to 0.01 per cent. of carbon; the figure for manganese is taken from Table VII. Fig. 3 is plotted from the last column, but the data are combined to rectify the lines.

Limits of Carbon.	Number of Heats.	Chemical Composition.				Ultimate Strength.	
		Carbon.	Phosphorus.	Manganese.	Sulphur.	Actual Records.	After Deducting for Carbon and Manganese.
Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Pounds per Square Inch.	Pounds per Square Inch.
0.075 to 0.224	39	0.1491	0.0396	0.439	0.0539	59,994	44,616
	54	0.1524	0.0500	0.430	0.0559	61,038	45,438
	38	0.1504	0.0557	0.441	0.0568	61,595	46,063
	61	0.1528	0.0617	0.445	0.0588	62,633	46,813
	47	0.1540	0.0717	0.447	0.0623	63,292	47,328
0.225 to 0.374	46	0.3373	0.0331	0.514	0.0477	79,636	42,805
	53	0.3317	0.0438	0.537	0.0529	81,231	44,444
	44	0.3265	0.0523	0.527	0.0538	81,197	45,194
	49	0.3120	0.0626	0.537	0.0537	80,390	45,792
0.375 to 0.524	52	0.4413	0.0271	0.514	0.0437	90,413	42,270
	63	0.4424	0.0343	0.508	0.0461	91,180	43,138
	54	0.4366	0.0404	0.521	0.0494	92,215	44,320
	62	0.4235	0.0504	0.534	0.0526	91,370	44,517

EFFECT OF MANGANESE ON ACID STEEL.

First Method.—The heats were divided into "low" and "high" manganese in the same way and within the same carbon-limits as already described in the determination of phosphorus. The results as given in Table V show that the effect of one unit of manganese is greater as the carbon increases. The increase is not regular, but this is partly explained by the small number of heats in some of the groups. Combining the data so as to have three larger groups, and plotting the results, it was found that for each increase of 0.01 per cent. of carbon, the effect of 0.01 per cent. of manganese was 10 lb. more. That is to say, if 0.01 per cent. of manganese strengthens a steel of 0.2 per cent. carbon by 160 lb., it will strengthen a steel of 0.21 per cent. of carbon by 170 pounds. In the second method of determining manganese it is found that the increment is 8 lb., which agrees fairly well with this valuation.

TABLE V.—*Division of Heats to Determine the Effect of Manganese on Acid Steel.*

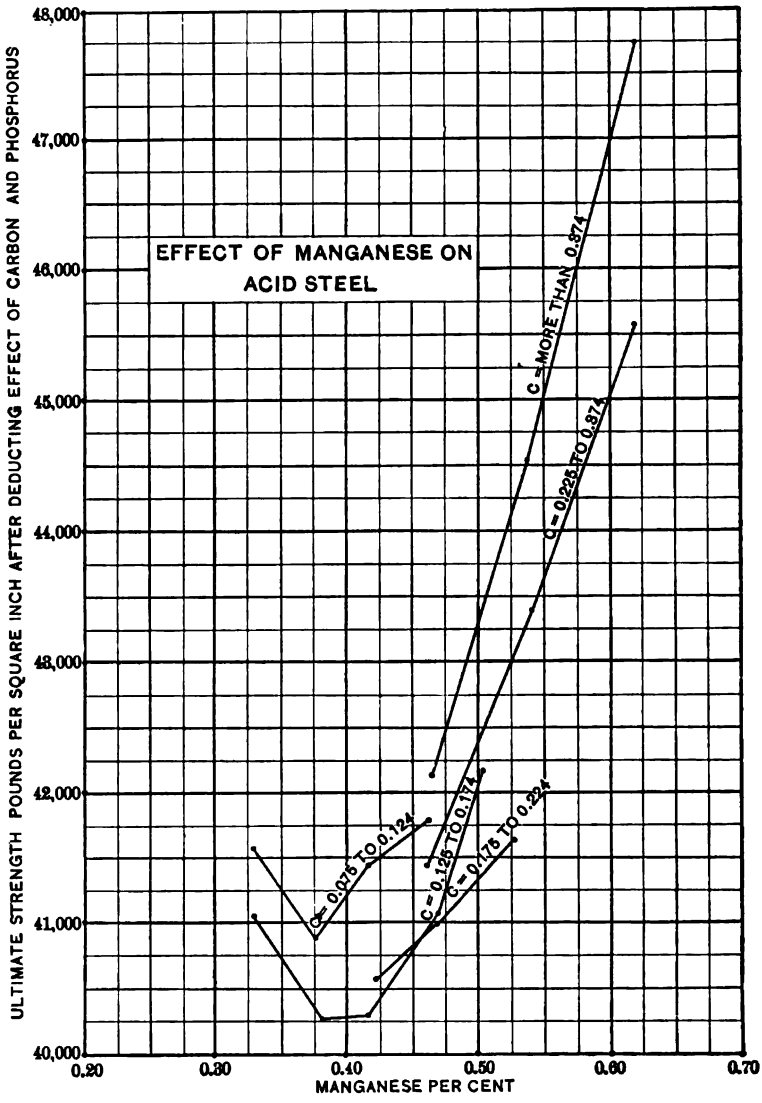
NOTE.—In the eighth column a value is given both to carbon and phosphorus of 1,000 lb. for 0.01 per cent.

Limits of Carbon.	Relative Manganese.	Number of Heats.	Chemical Composition.			Ultimate Strength.		
			Carbon.	Phosphorus.	Manganese.	Actual Records.	Difference Due to Manganese Alone.	Effect of 0.01 Per Cent. of Manganese.
Per Cent.			Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
	High.	27	0.1156	0.0564	0.440	58,847	.....	.....
0.075 to 0.124	Low..	23	0.1073	0.0524	0.370	57,031	.....	.....
	Diff...		0.0083	0.0040	0.070	1,816	586	84
	High.	63	0.1517	0.0572	0.476	62,008	.....	.....
0.125 to 0.174	Low..	68	0.1413	0.0562	0.402	60,142	.....	.....
	Diff...		0.0104	0.0010	0.074	1,866	726	98
	High.	27	0.1974	0.0590	0.514	66,936	.....	.....
0.175 to 0.224	Low..	31	0.2012	0.0567	0.440	66,698	.....	.....
	Diff...		0.0038	0.0023	0.074	238	388	52
	High.	10	0.2413	0.0551	0.519	70,602	.....	.....
0.225 to 0.274	Low..	12	0.2505	0.0574	0.456	70,850	.....	.....
	Diff..		0.0092	0.0023	0.063	−248	902	143
	High.	26	0.3048	0.0524	0.568	79,926	.....	.....
0.275 to 0.324	Low..	24	0.3083	0.0425	0.486	78,118	.....	.....
	Diff...		0.0035	0.0099	0.082	1,808	1,168	142
	High.	59	0.3513	0.0476	0.582	84,670	.....	.....
0.325 to 0.374	Low..	61	0.3489	0.0458	0.493	81,569	.....	.....
	Diff...		0.0024	0.0018	0.089	3,101	2,661	299
	High.	53	0.3987	0.0405	0.556	88,554	.....	.....
0.375 to 0.424	Low..	50	0.4014	0.0394	0.478	85,675	.....	.....
	Diff...		0.0027	0.0011	0.078	2,879	3,039	390
	High.	45	0.4492	0.0383	0.560	94,174	.....	.....
0.425 to 0.474	Low..	41	0.4490	0.0368	0.476	91,465	.....	.....
	Diff...		0.0002	0.0015	0.084	2,709	2,539	302
	High.	22	0.4970	0.0374	0.557	100,097	.....	.....
0.475 to 0.524	Low..	20	0.4948	0.0352	0.477	96,163	.....	.....
	Diff...		0.0022	0.0022	0.080	3,934	3,494	437
	High.	4	0.5500	0.0383	0.518	104,248	.....	.....
0.525 to 0.574	Low..	4	0.5440	0.0325	0.473	100,445	.....	.....
	Diff...		0.0060	0.0058	0.045	3,803	2,623	583
	High.	3	0.5827	0.0377	0.533	109,945	.....	.....
0.575 to 0.624	Low..	3	0.5897	0.0283	0.453	104,850	.....	.....
	Diff...		0.0070	0.0094	0.080	5,095	4,855	607

Second Method.—The heats were divided according to their content of manganese in the same way as explained in the the second determination of phosphorus. The results as given in Table VI and in Fig. 3 show that when the manganese exceeds 0.4 per cent. each increase in that element raises the strength, while with a content below 0.4 per cent. the tensile strength increases as the manganese decreases. The number of observations of low-manganese acid steels is not sufficient to

prove this conclusively, but on another page it will be shown that in basic steel also, a decrease in the manganese-content below a certain point is not accompanied by a decrease in

FIG. 3.



EFFECT OF MANGANESE ON ACID STEEL.

strength. It is probable that low-manganese implies the presence of iron oxide and that this strengthens the steel much more than it is weakened by the decrease in manganese.

The lines in Fig. 3 show that each increase in manganese above 0.4 per cent. is accompanied by an increase in strength, but this increase is not the same with steels of different carbon. In steels containing more than 0.374 per cent. of carbon, each increase of 0.01 per cent. of manganese augments the tensile strength by about 440 lb. per sq. in. In Table VI it is shown that the average carbon of this group is about 0.44 per cent., and we thus determine that for a steel of 0.44 per cent. of carbon, the strengthening effect of 0.01 per cent. of manganese is about 440 lb. per sq. in. In the same way the line of next lower carbon shows that in steels of 0.33 per cent. of carbon, the strengthening effect is about 260 lb. per sq. in. The next three lines may be considered as a unit indicating that for steels of 0.155 per cent. of carbon, the strengthening effect is about 125 lb. per sq. in. Plotting these data it was found that the strengthening effect of each 0.01 per cent. of manganese above a content of 0.4 per cent. is 80 lb. per sq. in. for a steel of 0.1 per cent. of carbon, but that for each rise of 0.01 per cent. of carbon the strengthening effect is increased 8 pounds. Thus an increase in manganese from 0.4 to 0.41 per cent. in steel of 0.1 per cent. of carbon raises the strength 80 lb., but an increase in manganese from 0.4 to 0.41 per cent. in steel of 0.11 per cent. of carbon raises the strength 88 pounds. A continuation of the line thus plotted gave zero-effect for zero carbon. With basic steel it will appear that a different value was obtained for a starting point and a different value for the increment. The law of variation in the effect of manganese upon acid steels is shown in Table VII.

Effect of Sulphur on Acid Steel.

First Method.—The heats were divided into high- and low-sulphur as shown in Table VIII following the same system as used with manganese and phosphorus. The results give a value of minus 100 lb. for 0.01 per cent. of sulphur, indicating that 0.01 per cent. weakens the steel by 100 lb. per sq. in.

Second Method.—The heats were classified according to their sulphur-content, the results being shown in Table IX and in Fig. 4. The second method corroborates the first in showing that sulphur has little influence upon the strength of acid steel.

TABLE VI.—*Classification of Acid Heats According to Content of Manganese.*

NOTE.—In the last column both carbon and phosphorus are valued at 1,000 lb. for 0.01 per cent.

Limits of Carbon.	Limits of Manganese.	Number of Heats.	Chemical Composition.				Ultimate Strength.	
			Carbon.	Phosphorus.	Manganese.	Sulphur.	Actual Records.	After Deducting for Carbon and Phosphorus.
Per Cent.	Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
0.075 to 0.124	0.30 to 0.35	6	0.1052	0.0548	0.330	0.0560	57,558	41,558
	0.36 to 0.39	12	0.1117	0.0500	0.377	0.0576	57,047	40,877
	0.40 to 0.44	20	0.1110	0.0564	0.416	0.0589	58,173	41,433
	0.45 to 0.49	11	0.1168	0.0568	0.462	0.0636	59,135	41,775
0.125 to 0.174	0.30 to 0.35	2	0.1330	0.0585	0.330	0.0550	60,200	41,050
	0.36 to 0.39	19	0.1354	0.0538	0.381	0.0564	59,189	40,269
	0.40 to 0.44	55	0.1459	0.0569	0.417	0.0579	60,560	40,280
	0.45 to 0.49	41	0.1477	0.0564	0.470	0.0595	61,483	41,073
	0.50 to 0.59	14	0.1608	0.0601	0.503	.....	64,253	42,163
0.175 to 0.224	0.40 to 0.44	16	0.2004	0.0562	0.422	0.0504	66,237	40,577
	0.45 to 0.49	23	0.2016	0.0587	0.468	0.0567	67,020	40,990
	0.50 to 0.59	19	0.1960	0.0579	0.527	.....	67,035	41,645
0.225 to 0.374	0.40 to 0.49	47	0.3127	0.0476	0.461	.....	77,471	41,441
	0.50 to 0.59	122	0.3305	0.0482	0.541	.....	81,257	43,387
	0.60 to 0.69	19	0.3413	0.0476	0.618	.....	84,463	45,573
Over 0.374	0.40 to 0.49	83	0.4495	0.0359	0.465	.....	90,680	42,140
	0.50 to 0.59	144	0.4387	0.0395	0.537	.....	92,365	44,545
	0.60 to 0.69	17	0.4461	0.0387	0.618	.....	96,218	47,738

TABLE VII.—*Effect of Manganese on Acid Steel.*

Carbon.	Manganese.										
	Per Cent. 0.40	Per Cent. 0.42	Per Cent. 0.44	Per Cent. 0.46	Per Cent. 0.48	Per Cent. 0.50	Per Cent. 0.52	Per Cent. 0.54	Per Cent. 0.56	Per Cent. 0.58	Per Cent. 0.60
Per Cent.		Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.
0.10	.....	160	320	480	640	800	960	1,120	1,280	1,440	1,600
0.15	.....	240	480	720	960	1,200	1,440	1,680	1,920	2,160	2,400
0.20	.....	320	640	960	1,280	1,600	1,920	2,240	2,560	2,880	3,200
0.25	.....	400	800	1,200	1,600	2,000	2,400	2,800	3,200	3,600	4,000
0.30	.....	480	960	1,440	1,920	2,400	2,880	3,360	3,840	4,320	4,800
0.35	.....	560	1,120	1,680	2,240	2,800	3,360	3,920	4,480	5,040	5,600
0.40	.....	640	1,280	1,920	2,560	3,200	3,840	4,480	5,120	5,760	6,400
0.45	.....	720	1,440	2,160	2,880	3,600	4,320	5,040	5,760	6,480	7,200
0.50	.....	800	1,600	2,400	3,200	4,000	4,800	5,600	6,400	7,200	8,000
0.55	.....	880	1,760	2,640	3,520	4,400	5,280	6,160	7,040	7,920	8,800
0.60	.....	960	1,920	2,880	3,840	4,800	5,760	6,720	7,680	8,640	9,600

TABLE VIII.—*Division of Heats to Determine the Effect of Sulphur on Acid Steel.*

NOTE.—In the eighth column both carbon and phosphorus are valued at 1,000 lb. for 0.01 per cent.; the figure for manganese is taken from Table VII.

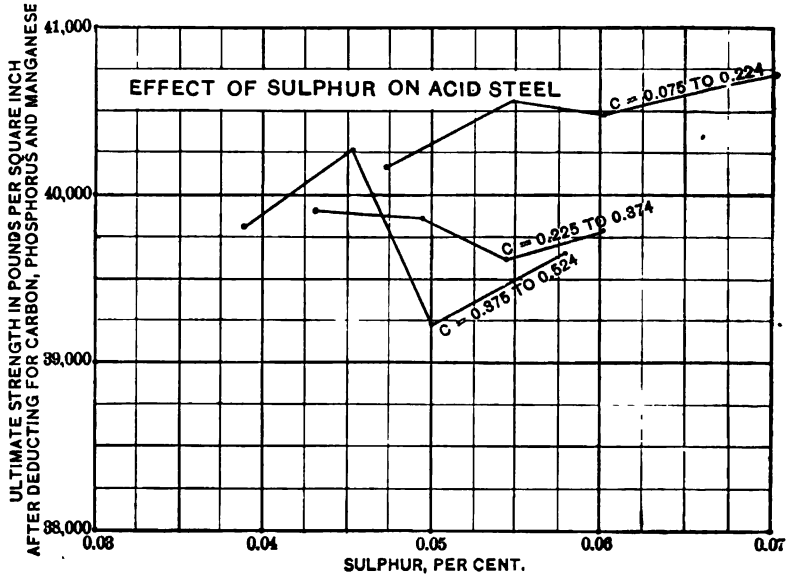
Limits of Carbon.	Relative Sulphur.	Chemical Composition.				Ultimate Strength.			Number of Heats.	Product of Last Two Columns.
		Carbon.	Phosphorus.	Manganese.	Sulphur.	Actual Records.	Difference Due to Sulphur Alone.	Effect of 0.01 Per Cent. of Sulphur.		
Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.		
	High.	0.1154	0.0587	0.424	0.0657	58,860	.....	.....	.....	.....
0.075 to 0.124	Low..	0.1079	0.0501	0.390	0.0523	57,093	.....	.....	.....	.....
	Diff...	0.0075	0.0086	0.034	0.0134	1,767	−143	−107	50	−5,350
	High.	0.1467	0.0600	0.439	0.0640	61,582	.....	.....	.....	.....
0.125 to 0.174	Low..	0.1485	0.0526	0.435	0.0509	60,398	.....	.....	.....	.....
	Diff...	0.0018	0.0074	0.004	0.0131	1,184	+576	+440	131	+57,640
	High.	0.2001	0.0604	0.488	0.0628	67,482	.....	.....	.....	.....
0.175 to 0.224	Low..	0.1987	0.0553	0.462	0.0490	66,181	.....	.....	.....	.....
	Diff...	0.0014	0.0051	0.026	0.0138	1,301	+235	+170	58	+9,860
	High.	0.2490	0.0596	0.486	0.0576	70,925	.....	.....	.....	.....
0.225 to 0.274	Low..	0.2435	0.0530	0.483	0.0488	70,550	.....	.....	.....	.....
	Diff...	0.0055	0.0066	0.003	0.0088	375	−895	−1,017	22	−22,374
	High.	0.3067	0.0526	0.538	0.0597	79,650	.....	.....	.....	.....
0.275 to 0.324	Low..	0.3062	0.0419	0.517	0.0475	78,363	.....	.....	.....	.....
	Diff...	0.0005	0.0107	0.021	0.0122	1,287	−337	−276	50	−13,800
	High.	0.3499	0.0496	0.546	0.0574	83,526	.....	.....	.....	.....
0.325 to 0.374	Low..	0.3503	0.0432	0.527	0.0460	82,615	.....	.....	.....	.....
	Diff...	0.0004	0.0064	0.019	0.0114	911	−221	−194	120	−23,280
	High.	0.3993	0.0437	0.527	0.0543	87,475	.....	.....	.....	.....
0.375 to 0.424	Low..	0.4007	0.0363	0.509	0.0424	86,830	.....	.....	.....	.....
	Diff...	0.0014	0.0074	0.018	0.0119	645	−531	−446	103	−45,938
	High.	0.4522	0.0418	0.529	0.0551	93,721	.....	.....	.....	.....
0.425 to 0.474	Low..	0.4462	0.0336	0.512	0.0413	92,082	.....	.....	.....	.....
	Diff...	0.0060	0.0082	0.017	0.0138	1,639	−393	−285	86	−24,510
	High.	0.4944	0.0380	0.518	0.0519	97,911	.....	.....	.....	.....
0.475 to 0.524	Low..	0.4975	0.0347	0.520	0.0426	98,536	.....	.....	.....	.....
	Diff...	0.0031	0.0033	0.002	0.0093	−625	−565	−608	42	−25,536
	High.	0.5421	0.0405	0.500	0.0510	103,665	.....	.....	.....	.....
0.525 to 0.574	Low..	0.5517	0.0302	0.490	0.0422	101,027	.....	.....	.....	.....
	Diff...	0.0096	0.0103	0.010	0.0088	2,638	+2,128	+2,418	8	+19,344
	High.	0.5897	0.0300	0.507	0.0503	108,550	.....	.....	.....	.....
0.575 to 0.624	Low..	0.5827	0.0360	0.480	0.0417	106,245	.....	.....	.....	.....
	Diff...	0.0070	0.0060	0.027	0.0086	2,305	+952	+1,107	6	+6,642
Total.....									676	−67,302
Average.....										−100

TABLE IX.—*Classification of Acid Heats According to Content of Sulphur.*

NOTE.—In the last column a value of 1,000 lb. is given to 0.01 per cent. of both carbon and phosphorus; the figure for manganese is taken from Table VII.

Limits of Carbon.	Number of Heats.	Chemical Composition.				Ultimate Strength.	
		Carbon.	Phosphorus.	Manganese.	Sulphur.	Actual Records.	After Deducting for Carbon, Phosphorus and Manganese.
Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Pounds per Square Inch.	Pounds per Square Inch.
0.075 to 0.224	58	0.1601	0.0519	0.425	0.0474	61,689	40,169
	68	0.1457	0.0546	0.444	0.0547	61,097	40,561
	61	0.1551	0.0581	0.448	0.0602	62,376	40,486
	52	0.1474	0.0621	0.444	0.0703	62,195	40,717
0.225 to 0.374	44	0.3345	0.0401	0.518	0.0431	80,478	39,903
	37	0.3288	0.0470	0.527	0.0495	80,798	39,865
	60	0.3256	0.0499	0.533	0.0544	80,670	39,609
	51	0.3203	0.0532	0.535	0.0612	80,582	39,776
0.375 to 0.524	63	0.4356	0.0330	0.514	0.0389	90,689	39,816
	45	0.4419	0.0367	0.511	0.0454	92,041	40,274
	64	0.4378	0.0392	0.515	0.0500	90,988	39,240
	59	0.4290	0.0449	0.536	0.0579	91,726	39,658

FIG. 4.

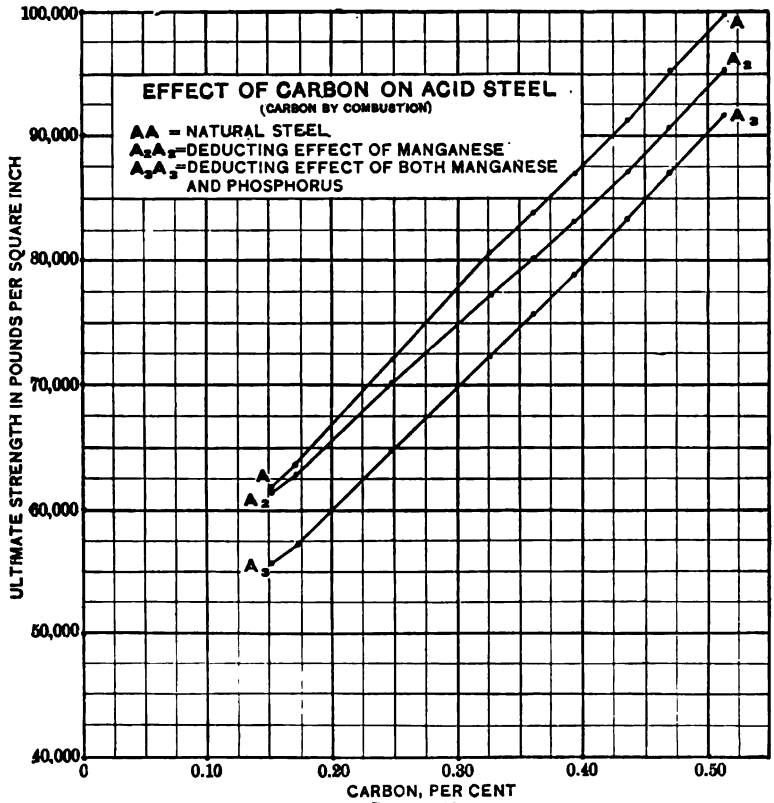


EFFECT OF SULPHUR ON ACID STEEL.

Effect of Carbon on Acid Steel.

Having found the effect of manganese and phosphorus, it becomes possible to correct the original line so as to determine the value of carbon. It so happens that the heats of higher carbon are at the same time of higher manganese. This makes a double correction, as an allowance must be made for the

Fig. 5.



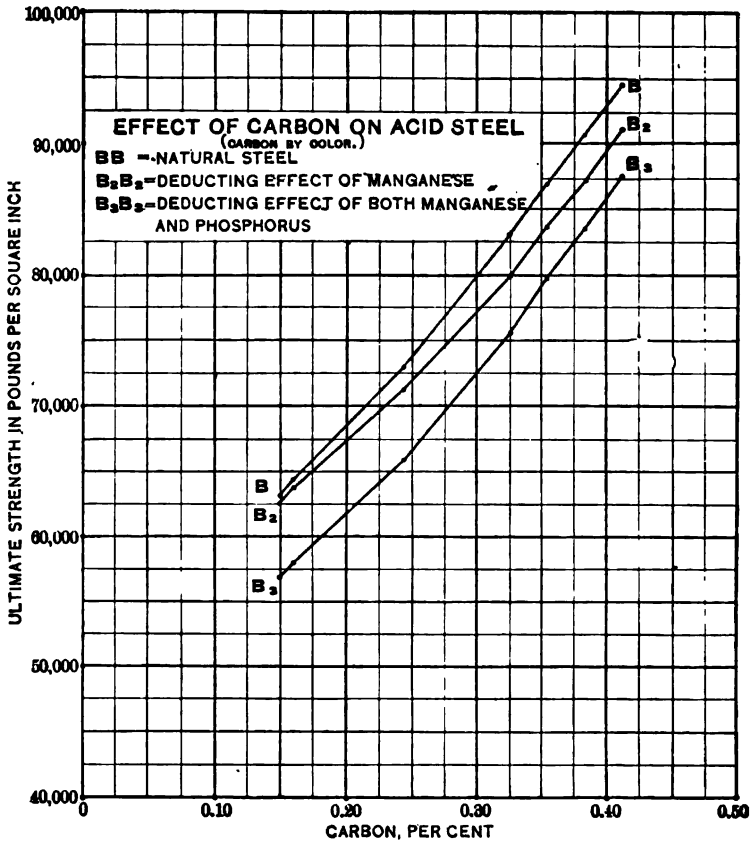
Effect of Carbon on Acid Steel (Carbon by Combustion).

greater amount of manganese and for the greater effect of this element in steels of higher carbon. The allowances are made in accordance with Table VII. The result is to drop the upper end of the line more than the lower and thereby decrease the angle which the line makes with the horizontal, this angle measuring the effect of carbon.

In allowing for phosphorus very little change is made in

this angle as the proportion of phosphorus is nearly the same in the low- and in the high-carbon steels, but the whole line is lowered, thereby giving a lower value for the point where the prolongation of the line intersects the ordinate of zero carbon. Table X gives the corrected values which are plotted in Figs. 5 and 6. The lines AA and BB are copied from Fig. 1; the

FIG. 6.



EFFECT OF CARBON ON ACID STEEL (CARBON BY COLOR).

lines A_2A_2 and B_2B_2 represent the allowance for manganese, and the lines A_3A_3 and B_3B_3 are corrected for both manganese and phosphorus. The line A_3A_3 indicates a value of 1,000 lb. for each 0.01 per cent. of carbon, when the combustion method is used, and it intersects the zero ordinate at 40,000 pounds. The line B_3B_3 indicates a value of 1,140 lb. for each 0.01 per cent. of

carbon, when the color method is used, and it intersects the zero-ordinate at 39,800 pounds.

TABLE X.—*Effect of Carbon on Acid Steel.*

NOTE.—In calculating the last column a value of 1,000 lb. is given to 0.01 per cent. of phosphorus; manganese is rated according to Table VII.

Class.	Chemical Composition.			Ultimate Strength.	
	Carbon.	Phosphorus.	Manganese.	Actual Records.	After Deducting for Phosphorus and Manganese.
	Per Cent.	Per Cent.	Per Cent.	Pounds per Square Inch.	Pounds per Square Inch.
Acid test-bars; carbon by combustion.	0.1520	0.0565	0.440	61,806	55,676
	0.1713	0.0570	0.453	63,637	57,216
	0.2486	0.0537	0.497	72,185	64,875
	0.3268	0.0480	0.529	80,626	72,472
	0.3609	0.0443	0.528	83,886	75,744
	0.3943	0.0419	0.526	87,155	78,996
	0.4357	0.0384	0.519	91,278	83,273
	0.4693	0.0371	0.518	95,052	86,917
	0.5130	0.0358	0.513	99,795	91,605
Acid test-bars; carbon by color.	0.1489	0.0562	0.443	63,072	56,936
	0.1600	0.0564	0.453	64,379	58,061
	0.2437	0.0541	0.491	72,980	65,823
	0.3255	0.0434	0.519	83,168	75,686
	0.3534	0.0403	0.515	87,012	79,762
	0.3827	0.0364	0.513	90,742	83,667
	0.4112	0.0351	0.506	94,689	87,702

EFFECT OF MANGANESE ON BASIC STEEL.

First Method.—The bars were divided into high- and low-manganese as shown in Table XI. The figures show an increasing value for manganese as the carbon increases, thereby agreeing with the work on acid steel. In the first group, composed of heats of very soft steel, the value of manganese is practically zero. This is the same thing as saying that the decrease in manganese-content from 0.408 per cent. to 0.118 per cent. did not decrease the strength, which is entirely in accord with the theory before advanced that iron oxide strengthens steel.

Second Method.—The bars were classified according to their content of manganese as shown in Table XII and in Fig. 7. The line of very low-carbon and low-manganese steels shows that in the absence of manganese the strength is raised by iron oxide or by some other agent. In steels of higher carbon less

TABLE XI.—*Division of Heats to Determine the Effect of Manganese on Basic Steel.*

NOTE.—In the eighth column a value of 770 lb. is given to 0.01 per cent. of carbon, and 1,000 lb. to 0.01 per cent. of phosphorus.

Limits of Carbon.	Relative Manganese.	Number of Heats.	Chemical Composition.			Ultimate Strength.		
			Carbon.	Phosphorus.	Manganese.	Actual Records.	Difference Due to Manganese Alone.	Effect of 0.01 Per Cent. of Manganese.
Per Cent.			Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
	High.	58	0.0622	0.0094	0.408	48,152	.....	.....
0.020 to 0.074	Low..	77	0.0322	0.0073	0.118	45,669	.....	.....
			0.0300	0.0021	0.290	2,483	−37	−1
	High.	64	0.0968	0.0086	0.485	50,726	.....	.....
0.075 to 0.124	Low..	61	0.0980	0.0083	0.357	49,445	.....	.....
			0.0012	0.0003	0.128	1,281	1,177	92
	High.	68	0.1555	0.0121	0.497	56,680	.....	.....
0.125 to 0.174	Low..	66	0.1488	0.0112	0.374	54,887	.....	.....
			0.0067	0.0009	0.123	1,793	1,003	82
	High.	120	0.2049	0.0130	0.535	62,370	.....	.....
0.175 to 0.224	Low..	126	0.2040	0.0097	0.411	60,162	.....	.....
			0.0009	0.0033	0.124	2,208	1,598	129
	High.	128	0.2495	0.0123	0.532	67,058	.....	.....
0.225 to 0.274	Low..	135	0.2474	0.0098	0.415	62,528	.....	.....
			0.0021	0.0025	0.117	4,530	3,896	333
	High.	60	0.2907	0.0123	0.525	69,593	.....	.....
0.275 to 0.324	Low..	65	0.2960	0.0090	0.408	67,144	.....	.....
			0.0053	0.0033	0.117	2,449	2,286	195
	High.	14	0.3414	0.0099	0.516	78,116	.....	.....
0.325 to 0.374	Low..	13	0.3411	0.0126	0.401	70,932	.....	.....
			0.0003	0.0027	0.115	2,184	2,171	189
	High.	6	0.3938	0.0150	0.583	82,695	.....	.....
0.375 to 0.424	Low..	5	0.3924	0.0080	0.398	73,741	.....	.....
			0.0014	0.0070	0.185	8,954	7,691	416

oxygen is present owing to the protecting power of carbon, and the decrease in strength with decrease in manganese holds good down to a content of 0.3 per cent. Considering only the lines representing steels with from 0.075 to 0.224 per cent. and with from 0.225 to 0.374 per cent. of carbon, and pursuing the same course of reasoning as explained in the valuation of manganese in acid steels, it appears that above the limit of 0.3 per cent. of manganese, the effect of each unit of that element is greater in the steels of higher carbon. In the acid steel the value at zero carbon was zero, the effect of 0.01 per cent. of manganese in a steel of 0.1 per cent. of carbon was 80 lb., and this effect increased 8 lb. with each rise of 0.01 per cent. of carbon.

TABLE XII.—*Classification of Basic Heats According to Content of Manganese.*

NOTE.—In the last column a value of 770 lb. is given to 0.01 per cent. of carbon, and 1,000 lb. to 0.01 per cent. of phosphorus.

Limits of Carbon.	Limits of Manganese.	Number of Heats.	Chemical Composition.			Ultimate Strength.	
			Carbon.	Phosphorus.	Manganese.	Actual Records.	After Deducting for Carbon and Phosphorus.
Per Cent.	Per Cent.		Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
Below 0.075	0.05 to 0.09	12	0.0297	0.0075	0.081	45,803	42,766
	0.10 to 0.14	56	0.0327	0.0073	0.120	45,674	42,426
	0.15 to 0.29	13	0.0388	0.0072	0.191	45,961	42,254
	0.30 to 0.39	16	0.0608	0.0097	0.354	48,034	42,390
	0.40 to 0.49	34	0.0632	0.0091	0.438	47,981	42,205
	0.50 to 0.59	4	0.0663	0.0133	0.508	51,133	44,698
0.075 to 0.224	0.20 to 0.29	7	0.1103	0.0079	0.259	50,056	40,773
	0.30 to 0.39	114	0.1458	0.0098	0.363	54,110	41,904
	0.40 to 0.49	242	0.1668	0.0099	0.441	57,036	43,203
	0.50 to 0.59	110	0.1744	0.0125	0.531	59,316	44,638
	0.60 to 0.69	26	0.1887	0.0154	0.622	61,862	45,793
0.225 to 0.374	0.30 to 0.39	61	0.2678	0.0089	0.365	63,858	42,349
	0.40 to 0.49	221	0.2689	0 0101	0.446	65,949	44,236
	0.50 to 0.59	102	0.2668	0.0130	0.532	67,565	45,723
	0.60 to 0.69	28	0.2695	0.0139	0.624	69,467	47,327

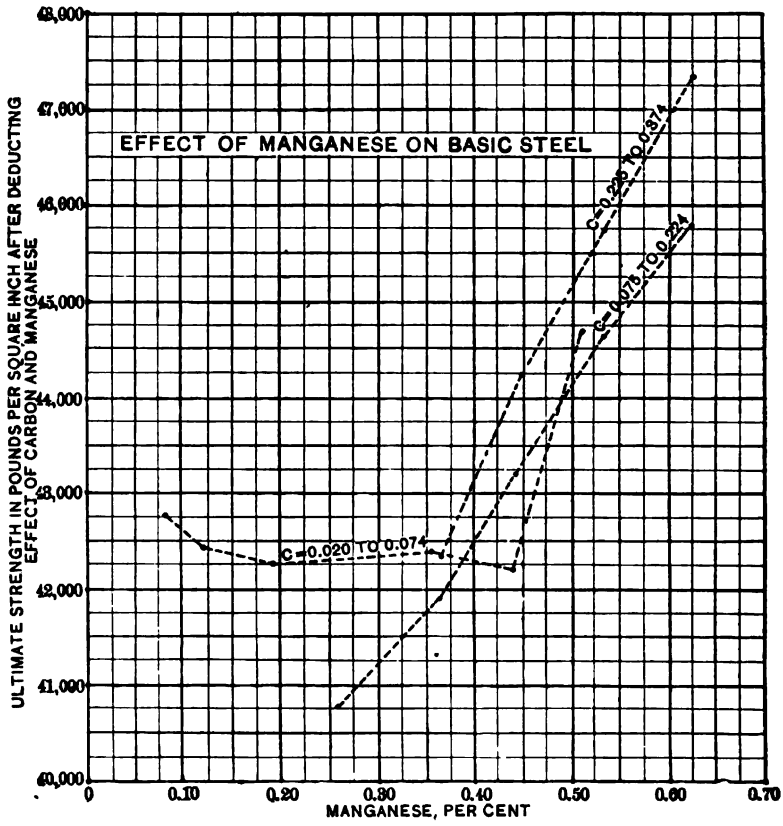
TABLE XIII.—*Effect of Manganese on Basic Steel.*

Carbon.	Manganese.						
	Per Cent. 0.30	Per Cent. 0.35	Per Cent. 0.40	Per Cent. 0.45	Per Cent. 0.50	Per Cent. 0.55	Per Cent. 0.60
Per Cent.		Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
0.05	.....	550	1,100	1,650	2,200	2,750	3,300
0.10	.....	650	1,300	1,950	2,600	3,250	3,900
0.15	.....	750	1,500	2,250	3,000	3,750	4,500
0.20	.....	850	1,700	2,550	3,400	4,250	5,100
0.25	.....	950	1,900	2,850	3,800	4,750	5,700
0.30	.....	1,050	2,100	3,150	4,200	5,250	6,300
0.35	.....	1,150	2,300	3,450	4,600	5,750	6,900
0.40	.....	1,250	2,500	3,750	5,000	6,250	7,500

In basic steel the value of 0.01 per cent. of manganese at zero carbon is 90 lb.; the effect per 0.01 per cent. of manganese

at 0.1 per cent. of carbon is 130 lb., and the increase in effect due to a rise of 0.01 per cent. of carbon is only 4 pounds. In the acid steel the base is 0.4 per cent. of manganese; in the basic steel it is 0.3 per cent. The results are tabulated in Table XIII.

FIG. 7.



EFFECT OF MANGANESE ON BASIC STEEL.

EFFECT OF SULPHUR ON BASIC STEEL.

First Method.—The heats were divided into high- and low-sulphur, as shown in Table XIV, and the results indicate that 0.01 per cent. of sulphur strengthens steel by 58 lb. per sq. in. In the acid steel the same method of analysis showed a weakening effect of 100 pounds. In either case the value is too small to be important.

TABLE XIV.—*Division of Heats to Determine the Effect of Sulphur on Basic Steel.*

NOTE.—In the eighth column a value of 770 lb. is given to 0.01 per cent. of carbon, and 1,000 lb. to 0.01 per cent. of phosphorus; manganese is rated as shown in Table XIII.

Limits of Carbon.	Relative Sulphur.	Chemical Composition.				Ultimate Strength.			Number of Heats.	Product of Last Two Columns.
		Carbon.	Phosphorus.	Manganese.	Sulphur.	Actual Records.	Difference Due to Sulphur Alone.	Effect of 0.01 Per Cent. of Sulphur.		
Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Pounds per Sq.Inch.	Pounds per Sq.Inch.	Pounds per Sq.Inch.		
	High.	0.0545	0.0091	0.323	0.0353	47,512	.....	.....	.....	.....
0.020 to 0.074	Low..	0.0361	0.0074	0.166	0.0240	46,000	.....	*Note	.....	.....
	Diff...	0.0184	0.0017	0.157	0.0113	1,512	—74	—65	135	—8,775
	High.	0.1012	0.0088	0.417	0.0452	51,070	.....	.....	.....	.....
0.075 to 0.124	Low..	0.0936	0.0081	0.428	0.0267	49,116	.....	.....	.....	.....
	Diff...	0.0076	0.0007	0.011	0.0185	1,954	+1,442	+779	125	+97,375
	High.	0.1505	0.0133	0.428	0.0563	56,196	.....	.....	.....	.....
0.125 to 0.174	Low..	0.1540	0.0099	0.446	0.0342	55,394	.....	.....	.....	.....
	Diff...	0.0035	0.0034	0.018	0.0221	802	+1,002	+453	133	+60,250
	High.	0.2038	0.0124	0.479	0.0553	61,236	.....	.....	.....	.....
0.175 to 0.224	Low..	0.2050	0.0104	0.465	0.0356	61,247	.....	.....	.....	.....
	Diff...	0.0012	0.0020	0.014	0.0197	—11	—357	—181	246	—44,522
	High.	0.2475	0.0132	0.487	0.0517	65,016	.....	.....	.....	.....
0.225 to 0.274	Low..	0.2493	0.0089	0.457	0.0331	64,452	.....	.....	.....	.....
	Diff...	0.0018	0.0043	0.030	0.0186	564	—297	—160	263	—42,080
	High.	0.2922	0.0120	0.477	0.0447	68,771	.....	.....	.....	.....
0.275 to 0.324	Low..	0.2947	0.0092	0.451	0.0323	67,875	.....	.....	.....	.....
	Diff...	0.0025	0.0028	0.026	0.0124	896	+273	+220	125	+27,500
	High.	0.3386	0.0128	0.450	0.0414	71,525	.....	.....	.....	.....
0.325 to 0.374	Low..	0.3441	0.0093	0.472	0.0294	72,647	.....	.....	.....	.....
	Diff...	0.0055	0.0035	0.022	0.0120	—1,122	—551	—459	27	—12,392
	High.	0.3950	0.0145	0.497	0.0397	78,400	.....	.....	.....	.....
0.375 to 0.424	Low..	0.3910	0.0086	0.502	0.0308	78,895	.....	.....	.....	.....
	Diff...	0.0040	0.0059	0.005	0.0089	—495	—1,270	—1,427	11	—15,697
Total.....									1,065	+61,659
Average.....										+58

*NOTE.—In calculating the difference due to sulphur in the group of very low carbon steels, no allowance is made for the difference in manganese, as a decrease in manganese to a content of 0.166 per cent. is not necessarily followed by a decrease in strength.

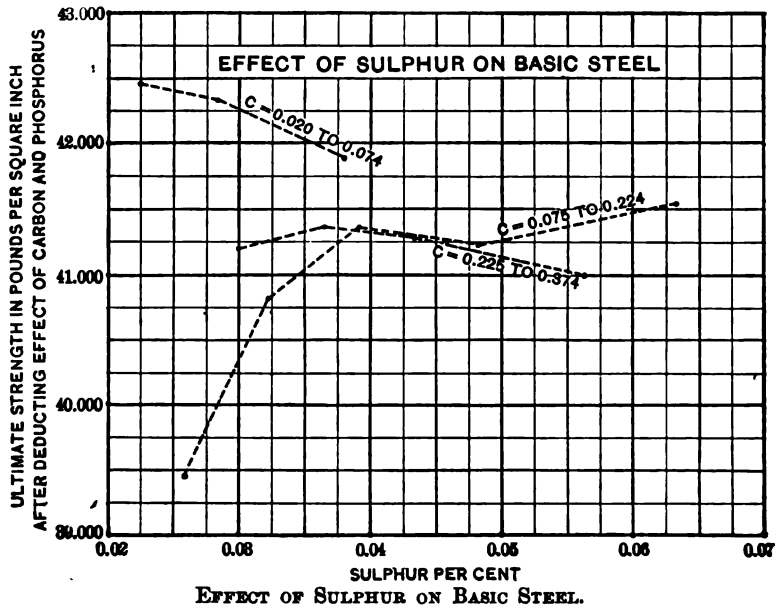
Second Method.—The heats were classified according to their sulphur-content, as shown in Table XV and in Fig. 8. The lines are irregular and indeterminate, indicating a very small value for this element.

TABLE XV.—*Classification of Basic Heats According to Content of Sulphur.*

NOTE.—In the last column a value of 770 lb. is given to 0.01 per cent. of carbon, and 1,000 lb. to 0.01 per cent. of phosphorus; manganese is rated as shown in Table XIII.

Limits of Carbon.	Number of Heats.	Chemical Composition.				Ultimate Strength.	
		Carbon.	Phosphorus.	Manganese.	Sulphur.	Actual Records.	After Deducting for Carbon, Phosphorus and Manganese.
Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Pounds per Square Inch.	Pounds per Square Inch.
Below 0.075	45	0.0361	0.0074	0.162	0.0225	45,978	42,458
	46	0.0418	0.0077	0.212	0.0283	46,337	42,348
	44	0.0575	0.0096	0.356	0.0380	47,922	41,896
0.075 to 0.224	74	0.1225	0.0078	0.434	0.0258	51,524	39,462
	103	0.1571	0.0089	0.444	0.0322	56,027	40,822
	112	0.1786	0.0114	0.466	0.0391	58,944	41,363
	105	0.1790	0.0115	0.461	0.0482	58,767	41,226
	110	0.1696	0.0129	0.441	0.0632	58,129	41,552
0.225 to 0.374	115	0.2754	0.0083	0.453	0.0298	66,333	41,206
	113	0.2693	0.0097	0.458	0.0365	66,194	41,360
	89	0.2679	0.0114	0.464	0.0434	66,307	41,292
	98	0.2582	0.0149	0.504	0.0563	66,334	41,005

FIG. 8.



EFFECT OF SULPHUR ON BASIC STEEL.

Effect of Carbon on Basic Steel.

The effect of carbon was found, as in the case of acid steels, by allowing for phosphorus and manganese in the groups given in Table II. The data are given in Table XVI and in Figs. 9 and 10. The line D_2D_3 indicates a value of 770 lb. for each 0.01 per cent. of carbon when the combustion method is used and it intersects the zero-ordinate at 41,500 pounds.

The line E_2E_3 indicates a value of 820 lb. for each 0.01 per cent. of carbon when the color-method is used, and it intersects the zero-ordinate at 42,000 pounds.

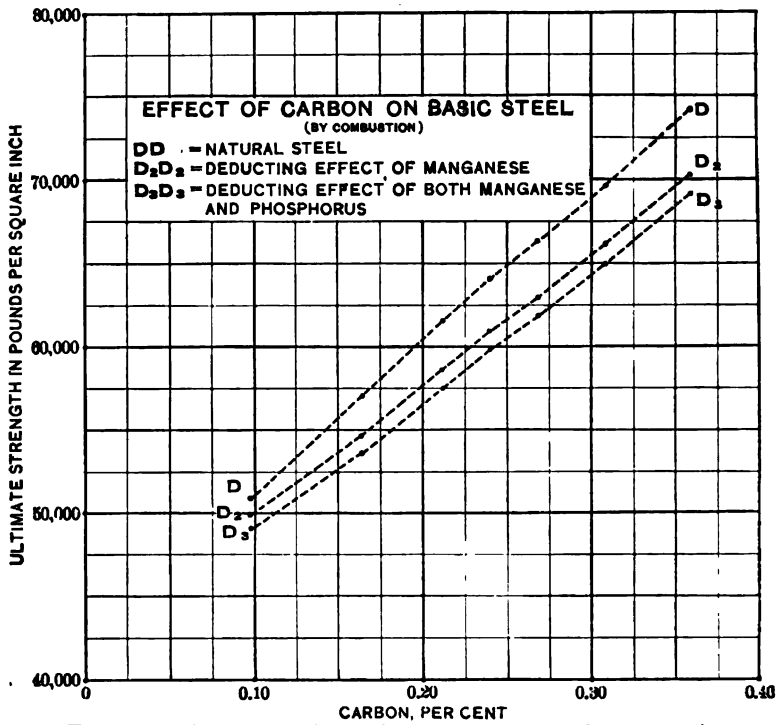
Table XVI.—*Effect of Carbon on Basic Steel.*

Note.—In calculating the last column a value of 1,000 lb. is given to 0.01 per cent. of phosphorus; the manganese is rated as shown in Table XIII.

Class.	Chemical Composition.			Ultimate Strength.	
	Carbon.	Phosphorus.	Manganese.	Actual Records.	After Deducting for Phosphorus and Manganese.
	Per Cent.	Per Cent.	Per Cent.	Pounds per Square Inch.	Pounds per Square Inch.
Basic test-bars; carbon by combustion.	0.0978	0.0094	0.366	50,834	49,036
	0.1639	0.0107	0.450	57,001	53,621
	0.2115	0.0113	0.465	61,502	57,501
	0.2403	0.0110	0.471	64,086	59,805
	0.2681	0.0109	0.470	66,297	61,841
	0.3081	0.0108	0.466	69,626	64,994
	0.3582	0.0113	0.469	74,203	69,118
Basic test-bars; carbon by color.	0.1010	0.0101	0.384	52,540	50,438
	0.1688	0.0116	0.458	59,739	56,083
	0.2036	0.0118	0.466	62,750	58,748
	0.2260	0.0118	0.468	64,839	60,601
	0.2564	0.0117	0.469	66,830	62,381

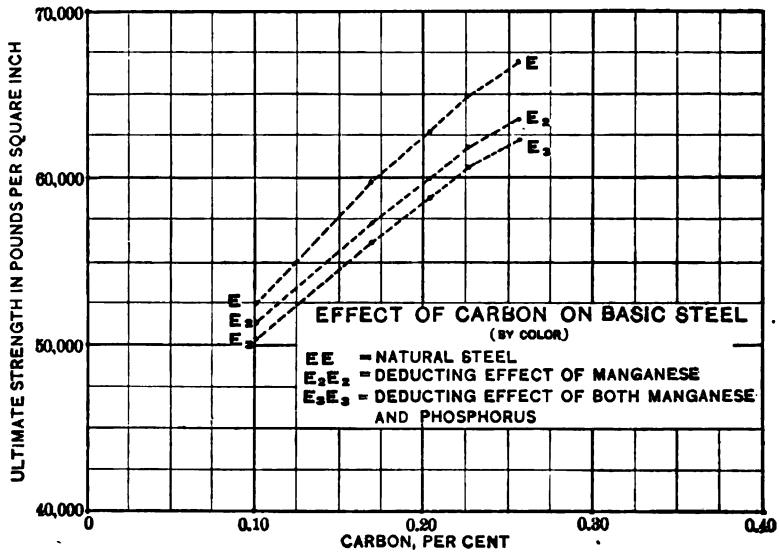
It has already been explained, and is shown by Fig. 9, that any changes in the value of manganese affects the tangent of the carbon-line, thereby affecting the value found for a unit of that element; and as manganese has been given a slightly higher value in basic than in acid steel, it would naturally follow that the result for carbon would be lower in the basic than in the acid steel. To find how much this change in the value of manganese affected the carbon determination, the experiment was tried of correcting the line of basic, according to the values of manganese found for acid steel. The result showed a value of 810 lb. for 0.01 per cent. of carbon instead of 770 lb., as found

FIG. 9.



EFFECT OF CARBON ON BASIC STEEL (CARBON BY COMBUSTION).

FIG. 10.



EFFECT OF CARBON ON BASIC STEEL (CARBON BY COLOR).

by the above special investigation. Inasmuch as the acid steel gave a value for carbon of 1,000 lb. per unit of 0.01 per cent., and as the basic steel gives 810 lb. when calculated by the acid formula and 770 lb. by its own formula, it would seem certain that a unit of carbon has much less effect upon basic than upon acid steel.

The Application of the Formulæ.

Table XVII shows the result of comparing the actual strength of the steels under consideration with the strength as calculated from the formulæ just given. For this purpose the heats were grouped according to carbon and then subdivided according to manganese. No heats were put together that varied more than 0.05 per cent. in carbon, or more than 0.1 per cent. in manganese. For instance, a group might include a heat containing 0.1 per cent. of carbon and 0.3 per cent. of manganese, and another heat containing 0.149 per cent. of carbon and 0.399 per cent. of manganese, but any heat of higher or lower carbon, or of higher or lower manganese than these extremes, would fall into another group. Inasmuch as the phosphorus did not vary through wide limits in any of the steels, each group may be looked upon as composed of heats that are practically alike in chemical composition, and which may properly be averaged to eliminate accidental errors.

In some of the subdivisions the number of heats is so small that these errors cloud the result. Especially in the steels of higher carbon it is desirable to have a large number of heats in the average, as it is difficult to get uniform results on a testing-machine under usual working conditions when the bar has a strength of over 90,000 lb. per sq. in., and unfortunately it is in these high steels and particularly in the groups with an unusual content of manganese that only a small number of heats were on record. There are accordingly several instances where these small groups show a considerable difference between the actual and the calculated strength, but there seems to be no rule as to the difference, as other groups, either large or small, of the same class of steels give satisfactory results.

It is of course a matter of opinion as to what constitutes a fair agreement between the actual and the calculated strengths, but in the following comparison it will be assumed that the

TABLE XVII.—*Comparison of the Actual Ultimate Strength of Certain Groups of Steel with the Strength as Calculated from the Following Formulæ:*

Acid steel: 40,000 + 1,000 C + 1,000 P + x Mn = ultimate strength.
Basic steel: 41,500 + 770 C + 1,000 P + y Mn = ultimate strength.

Value of x as per Table VII; value of y as per Table XIII.

Heavy type denote that the difference between the actual and calculated strengths is over 1,500 pounds.

Limits of Carbon.	Limits of Manganese.	Number of Heats.	Chemical Composition.			Ultimate Strength.		
			Carbon.	Phosphorus.	Manganese.	Actual Records.	By Formula.	Difference.
Per Cent.	Per Cent.		Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
Acid steel:								
0.075 to 0.124	0.30 to 0.39	18	0.1095	0.0517	0.361	57,217	56,120	—1,097
	0.40 to 0.49	31	0.1131	0.0566	0.432	58,414	57,258	—1,156
	0.50 to 0.59	1	0.1130	0.0440	0.500	56,745	56,600	—145
0.125 to 0.174	0.30 to 0.39	21	0.1352	0.0542	0.377	59,285	58,940	—345
	0.40 to 0.49	96	0.1466	0.0567	0.440	60,954	60,794	—160
	0.50 to 0.59	14	0.1608	0.0601	0.513	64,253	63,536	—717
0.175 to 0.224	0.40 to 0.49	39	0.2011	0.0577	0.449	66,698	66,664	—34
	0.50 to 0.59	19	0.1960	0.0579	0.527	67,035	67,371	+336
0.225 to 0.274	0.30 to 0.39	1	0.2340	0.0550	0.390	68,460	68,900	+440
	0.40 to 0.49	11	0.2520	0.0576	0.462	71,068	72,200	+1,132
	0.50 to 0.59	10	0.2413	0.0551	0.519	70,602	71,925	+1,323
0.275 to 0.324	0.40 to 0.49	14	0.3093	0.0446	0.469	78,200	77,101	—1,099
	0.50 to 0.59	32	0.3066	0.0485	0.541	79,167	78,950	—217
	0.60 to 0.69	3	0.2863	0.0497	0.610	80,223	78,456	1,767
	0.70 to 0.79	1	0.3240	0.0560	0.723	84,100	86,320	—2,220
0.325 to 0.374	0.30 to 0.39	1	0.3490	0.0340	0.300	81,650	78,300	—3,350
	0.40 to 0.49	22	0.3452	0.0446	0.455	80,208	80,498	+290
	0.50 to 0.59	80	0.3512	0.0472	0.544	83,425	83,872	+447
	0.60 to 0.69	16	0.3516	0.0472	0.619	85,258	86,012	+754
	0.70 to 0.79	1	0.3440	0.0460	0.700	86,840	87,180	+340
0.375 to 0.424	0.40 to 0.49	34	0.4009	0.0377	0.464	85,205	85,908	+703
	0.50 to 0.59	63	0.3996	0.0410	0.537	87,880	88,444	+564
	0.60 to 0.69	6	0.3993	0.0425	0.622	90,598	91,284	+686
0.425 to 0.474	0.40 to 0.49	27	0.4481	0.0363	0.462	90,950	90,672	—278
	0.50 to 0.59	53	0.4515	0.0382	0.539	93,760	93,974	+214
	0.60 to 0.69	6	0.4332	0.0378	0.617	93,805	94,587	+782
0.475 to 0.524	0.30 to 0.39	1	0.4770	0.0330	0.380	90,775	91,000	+225
	0.40 to 0.49	12	0.4955	0.0340	0.468	95,745	95,643	—102
	0.50 to 0.59	25	0.4961	0.0376	0.533	98,699	98,637	—62
	0.60 to 0.69	4	0.5010	0.0365	0.617	104,550	102,430	—2,120
0.525 to 0.574	0.40 to 0.49	6	0.5463	0.0303	0.478	100,718	101,061	+343
	0.50 to 0.59	2	0.5490	0.0505	0.545	107,230	106,330	—900
0.575 to 0.624	0.40 to 0.49	4	0.5887	0.0312	0.462	105,131	104,904	—227
	0.50 to 0.59	1	0.5770	0.0430	0.510	112,760	107,071	—5,689
	0.60 to 0.69	1	0.5850	0.0300	0.600	111,100	110,860	—240

Limits of Carbon.	Limits of Manganese.	Number of Heats.	Chemical Composition.			Ultimate Strength.		
			Carbon.	Phosphorus.	Manganese.	Actual Records.	By Formula.	Difference.
Per Cent.	Per Cent.		Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
Basic steel:								
0.020 to 0.074	0.00 to 0.09	12	0.0297	0.0075	0.081	45,803	44,537	—1,266
	0.10 to 0.19	65	0.0326	0.0073	0.125	45,645	44,740	—905
	0.20 to 0.29	4	0.0543	0.0073	0.263	47,094	46,411	—683
	0.30 to 0.39	16	0.0608	0.0097	0.354	48,034	47,767	—267
	0.40 to 0.49	34	0.0632	0.0091	0.438	47,981	48,849	+868
	0.50 to 0.59	4	0.0663	0.0133	0.508	51,133	50,389	—744
0.075 to 0.124	0.10 to 0.19	1	0.0990	0.0080	0.160	45,780	49,923	+4,143
	0.20 to 0.29	6	0.0993	0.0078	0.262	49,378	49,926	+548
	0.30 to 0.39	42	0.0983	0.0086	0.363	49,683	50,748	+1,065
	0.40 to 0.49	53	0.0955	0.0083	0.438	49,667	51,477	+1,810
	0.50 to 0.59	21	0.0998	0.0089	0.539	51,900	53,182	+1,282
	0.60 to 0.69	2	0.0950	0.0085	0.660	55,773	54,345	—1,428
0.125 to 0.174	0.10 to 0.19	1	0.1370	0.0070	0.160	52,295	52,749	+454
	0.30 to 0.39	41	0.1486	0.0107	0.359	54,738	54,897	+159
	0.40 to 0.49	64	0.1531	0.0114	0.445	55,800	56,596	+796
	0.50 to 0.59	24	0.1549	0.0130	0.535	57,050	58,300	+1,250
	0.60 to 0.69	3	0.1657	0.0213	0.640	59,943	61,693	+1,750
0.175 to 0.224	0.20 to 0.29	1	0.1760	0.0080	0.240	54,120	55,852	+1,732
	0.30 to 0.39	31	0.2064	0.0104	0.367	59,276	59,611	+335
	0.40 to 0.49	125	0.2040	0.0098	0.441	60,752	60,670	—82
	0.50 to 0.59	65	0.2059	0.0135	0.527	62,547	62,698	+151
	0.60 to 0.69	21	0.2009	0.0152	0.616	62,716	63,987	+1,271
	0.70 to 0.79	3	0.2050	0.0087	0.713	65,507	65,424	—83
0.225 to 0.274	0.20 to 0.29	1	0.2300	0.0070	0.260	61,090	59,909	—1,181
	0.30 to 0.39	39	0.2458	0.0079	0.365	62,185	62,463	+278
	0.40 to 0.49	137	0.2489	0.0105	0.451	64,425	64,644	+219
	0.50 to 0.59	66	0.2490	0.0132	0.529	66,107	66,436	+329
	0.60 to 0.69	18	0.2495	0.0141	0.627	67,048	68,465	+1,417
	0.70 to 0.79	1	0.2740	0.0140	0. 10	74,970	72,362	—2,608
	0.90 to 0.99	1	0.2280	0.0150	0. 40	67,595	72,395	+4,800
0.275 to 0.324	0.30 to 0.39	18	0.2986	0.0085	0.366	65,920	66,753	+833
	0.40 to 0.49	70	0.2937	0.0098	0.440	67,888	68,063	+175
	0.50 to 0.59	29	0.2907	0.0128	0.540	69,725	70,202	+477
	0.60 to 0.69	8	0.2900	0.0142	0.621	72,402	71,991	—411
0.325 to 0.374	0.30 to 0.39	4	0.3443	0.0200	0.355	70,954	71,286	+332
	0.40 to 0.49	14	0.3396	0.0086	0.437	71,170	71,660	+490
	0.50 to 0.59	7	0.3354	0.0114	0.524	72,365	73,572	+1,207
	0.60 to 0.69	2	0.3675	0.0105	0.610	79,515	78,286	—1,229
0.375 to 0.424	0.30 to 0.39	2	0.3830	0.0080	0.355	73,620	73,154	—466
	0.40 to 0.49	5	0.3936	0.0102	0.448	75,107	76,555	+1,448
	0.50 to 0.59	1	0.3860	0.0110	0.500	79,750	77,280	—2,470
	0.60 to 0.69	2	0.4065	0.0220	0.645	88,545	83,832	—4,713
	0.70 to 0.79	1	0.3920	0.0080	0.750	85,260	83,732	—1,528

results of the formulæ should be within 1,500 lb. of the records of the testing-machine. In the acid steels there are 12 groups containing less than 5 heats each. In 7 of these the cal-

culated strength agrees with the actual strength within 1,500 pounds. In 5 groups the difference is over 1,500 pounds. In the basic steel there are 17 groups containing less than 5 heats and 9 of these agree within 1,500 pounds. Eight groups show a difference greater than this amount. Taking both acid and basic steels, out of 29 " small " groups 16 are correct, and of the 13 that are beyond the limit 9 are single heats, most of them being steel of moderately high carbon.

In the acid steel there are 23 groups containing over 4 heats each, and all of them are within the limit of 1,500 lb., only 5 having an error exceeding 1,000 pounds. In the basic steel there are 26 groups with over 4 heats and 25 are within 1,500 lb., and 17 within 1,000 pounds. There is 1 group of 53 heats, averaging about 0.1 per cent. of carbon, which shows an error of + 1,810 pounds. Putting aside mathematical errors which can hardly be present in this investigation (owing to repeated checking of the totals at each separate rearrangement), it may appear probable that this group contains some abnormal bars, and it may also appear possible that some of the other large groups show an agreement through the averaging of bars showing wide differences among themselves.

Table XVIII gives some information on this point. Every group in Table XVII comprising more than 50 heats and containing less than 0.225 per cent. of carbon is subdivided so as to have only one-half the former variation in manganese. Thus, if a group comprised heats ranging from 0.4 to 0.49 per cent. of manganese, it is subdivided into one group ranging from 0.4 to 0.44 per cent., and another from 0.45 to 0.49 per cent. If the original group were an average of unlike units, it is probable that the fact would be made manifest by a wide difference between the two parts, but in no case is such a difference discernible.

In the case of the one group composed of 53 heats before mentioned, a more extended analysis is given in Table XVIII. It has been divided into 10 parts, the first containing only those heats that contained 0.4 per cent. of manganese, the second those with 0.41 per cent. of manganese, and so on. The number of heats in some of the subdivisions is small and complete regularity could hardly be expected, but in these 10 subdivisions the smallest difference between the strength as calculated by the formula and the strength as found by the testing-machine, is

TABLE XVIII.—*Subdivision of the Groups in Table XVII that contain over Fifty Heats, and are below* 0.225 *per cent. of carbon, with special subdivision of the one large group showing a difference of more than* 1,500 *lb. between the actual and calculated strength.*

Limits of Carbon.	Limits of Manganese.	Number of Heats.	Chemical Composition.			Ultimate Strength.		
			Carbon.	Phosphorus.	Manganese.	Actual Records.	By Formula.	Difference.
Per Cent.	Per Cent.		Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
Acid steel:								
0.125 to 0.174	0.40 to 0.44	55	0.1459	0.0569	0.417	60,560	60,484	—76
	0.45 to 0.49	41	0.1477	0.0564	0.470	61,483	61,250	—233
Basic steel:								
0.020 to 0.074	0.10 to 0.14	56	0.0327	0.0073	0.120	45,674	44,748	—926
	0.15 to 0.19	9	0.0319	0.0071	0.159	45,458	44,666	—792
0.075 to 0.124	0.40 to 0.44	33	0.0961	0.0086	0.418	49,809	51,294	+1,485
	0.45 to 0.49	20	0.0946	0.0079	0.470	49,434	51,784	+2,350
	0.40	12	0.0963	0.0075	0.400	48,949	50,965	+2,016
	0.41	4	0.0888	0.0110	0.410	49,510	50,824	+1,314
	0.42	5	0.0946	0.0080	0.420	49,469	51,096	+1,627
	0.43	4	0.1012	0.0083	0.430	50,626	51,812	+1,186
	0.44	8	0.0980	0.0091	0.440	51,053	51,776	+723
	0.45	4	0.0870	0.0078	0.450	48,521	50,869	+2,348
	0.46	5	0.0922	0.0084	0.460	49,693	51,455	+1,762
	0.47	3	0.0833	0.0073	0.470	48,383	50,718	+2,335
	0.48	4	0.1135	0.0080	0.480	50,993	53,452	+2,459
	0.49	4	0.0948	0.0075	0.490	49,253	51,982	+2,729
0.125 to 0.174	0.40 to 0.44	32	0.1522	0.0114	0.418	55,495	56,129	+634
	0.45 to 0.49	32	0.1541	0.0114	0.473	56,265	57,102	+837
0.175 to 0.224	0.40 to 0.44	66	0.2036	0.0090	0.416	60,344	60,095	—249
	0.45 to 0.49	59	0.2046	0.0107	0.468	61,208	61,247	+39
	0.50 to 0.54	48	0.2063	0.0139	0.514	62,358	62,584	+226
	0.55 to 0.59	17	0.2049	0.0124	0.566	63,086	63,199	+113

+ 723 lb., and the greatest is + 2,729 lb., so that the deviation of this group from the general rule is not due to one or two abnormal bars. With this one exception, the cause of which remains unexplained, all the large groups show a difference of less than 1,500 lb. between the actual and the calculated strength, which is perhaps as close an agreement as could be expected.

A careful analysis was made to discover whether anything could be learned from the so-called errors. If, for instance, the groups of low carbon had shown a considerable and uniform minus error and the groups of high carbon had uniformly shown a similar plus error, then it would be probable that the

value of carbon was too high and the base too low. Investigation failed to show any regular law either for groups of high and low carbon, or for groups of high and low manganese. The one fact which appears to be true of both acid and basic steel is that the steels that are low in carbon and low in manganese are stronger than would be called for by the formula, and it seems probable that this is due to iron oxide.

The Value of Manganese.

From Table XVII may be obtained data which will corroborate the variable value assigned to manganese. The groups containing from 0.3 to 0.39 per cent. of manganese may be plotted, using as abscissas the percentage of carbon and as ordinates the ultimate strength. The groups containing from 0.4 to 0.49 per cent. of manganese furnish another line; those from 0.5 to 0.59 per cent. another; and those from 0.6 to 0.69 another. Owing to this subdivision many of the construction-points in these lines represent only a small number of heats and they have therefore been combined by groups of three as before explained. The result is shown in Table XIX. If the phosphorus in each group were constant, no allowance would have to be made for it, but since it varies considerably, the ultimate strength has been calculated to zero phosphorus, as shown in the last column of the table and as plotted in Figs. 11 and 12.

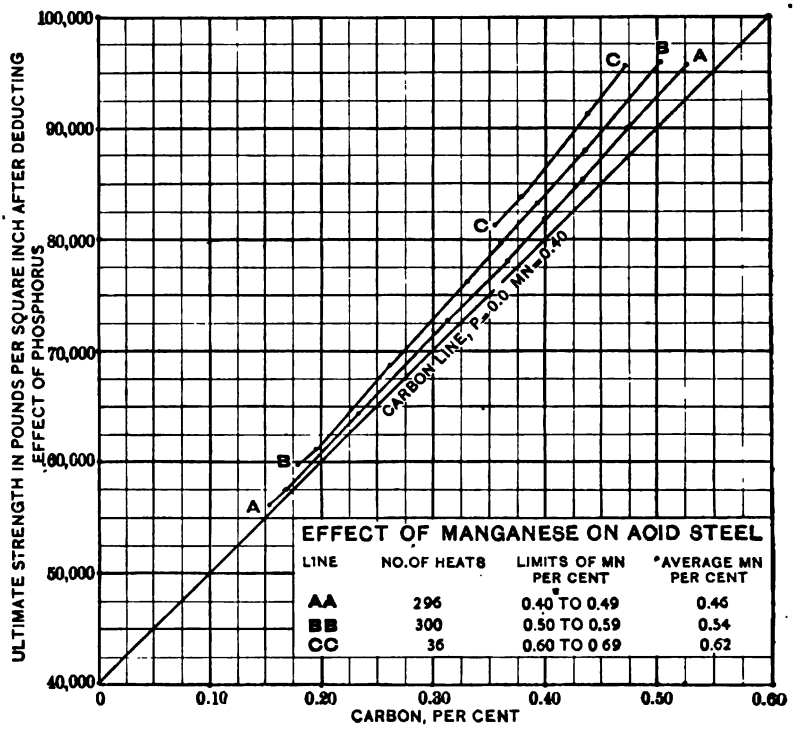
Fig. 11 shows that the line representing from 0.4 to 0.49 per cent. of manganese is slightly above the base-line of 0.4 of manganese, which is the limit already determined, below which a decrease in manganese does not weaken steel. The number of heats containing less than 0.4 per cent. of manganese in acid steel is small and they are confined almost entirely to the two low-carbon groups, so that no line has been plotted for them, but if these two groups be put upon the diagram it will be found that they are as strong as though the manganese were higher. Above the limit of 0.4 per cent., each increase in manganese raises the strength, and not only this, but the angle made with the horizontal is greater as the content of carbon increases. The lines tend to converge at the left of the figure at about 40,000 lb., which has been found to be the base for acid steel, and they tend to spread as they go to the right, which shows that the strengthening effect of manganese increases as

TABLE XIX.—*Classification of Groups in Table XVII According to Manganese and Combination in Units of Three.*

Kind of Steel and Composition.	Chemical Composition.			Ultimate Strength.	
	Carbon.	Phosphorus.	Manganese.	Actual Records.	Deducting Effect of Phosphorus.
	Per Cent.	Per Cent.	Per Cent.	Pounds per Square Inch.	Pounds per Square Inch.
Acid steel; Mn = 0.4 to 0.49 per cent. Line AA, Fig. 11.	0.1532	0.0569	0.441	61,840	56,180
	0.1691	0.0570	0.444	63,250	57,550
	0.2335	0.0548	0.456	69,963	64,483
	0.3127	0.0476	0.461	77,465	72,705
	0.3651	0.0412	0.462	82,238	78,118
	0.4004	0.0391	0.461	85,750	81,840
	0.4339	0.0366	0.464	89,060	85,400
	0.4738	0.0349	0.466	93,528	90,038
	0.5264	0.0325	0.470	98,860	95,610
Acid steel; Mn = 0.5 to 0.59 per cent. Line BB, Fig. 11.	0.1791	0.0584	0.520	65,590	59,750
	0.1951	0.0580	0.521	66,962	61,162
	0.2614	0.0525	0.533	73,985	68,735
	0.3305	0.0482	0.541	81,259	76,439
	0.3605	0.0452	0.541	84,250	79,730
	0.3938	0.0428	0.540	87,650	83,370
	0.4361	0.0394	0.537	92,030	88,090
	0.4679	0.0383	0.537	95,640	91,810
	0.5028	0.0387	0.533	99,800	95,930
Acid steel; Mn = 0.6 to 0.69 per cent. Line CC, Fig. 11.	0.3553	0.0463	0.619	85,940	81,310
	0.3793	0.0442	0.619	88,240	83,820
	0.4374	0.0392	0.619	95,280	91,360
	0.4716	0.0366	0.615	99,280	95,620
Basic steel; Mn = 0.3 to 0.39 per cent. Line AA, Fig. 12.	0.1131	0.0097	0.360	51,505	50,535
	0.1458	0.0098	0.363	54,107	53,127
	0.1989	0.0096	0.363	58,620	57,660
	0.2427	0.0089	0.366	61,900	61,010
	0.2678	0.0089	0.365	63,860	62,970
	0.3132	0.0104	0.363	67,397	66,357
Basic steel; Mn = 0.4 to 0.49 per cent. Line BB, Fig. 12.	0.1127	0.0098	0.441	51,920	50,940
	0.1668	0.0099	0.441	57,040	56,050
	0.2129	0.0104	0.446	61,340	60,300
	0.2415	0.0101	0.445	63,775	62,765
	0.2689	0.0101	0.446	65,955	64,945
	0.3065	0.0096	0.440	68,807	67,847
Basic steel; Mn = 0.5 to 0.59 per cent. Line CC, Fig. 12.	0.1240	0.0112	0.534	54,360	53,240
	0.1745	0.0125	0.531	59,315	58,065
	0.2163	0.0133	0.529	63,210	61,880
	0.2390	0.0133	0.530	65,315	63,985
	0.2667	0.0130	0.532	67,560	66,260
	0.3018	0.0125	0.536	70,495	69,245
Basic steel; Mn = 0.6 to 0.69 per cent. Line DD, Fig. 12.	0.1887	0.0154	0.622	61,864	60,324
	0.2192	0.0151	0.622	64,373	62,863
	0.2347	0.0146	0.621	66,020	64,560
	0.2695	0.0139	0.624	69,465	68,075
	0.3223	0.0149	0.623	76,278	74,788

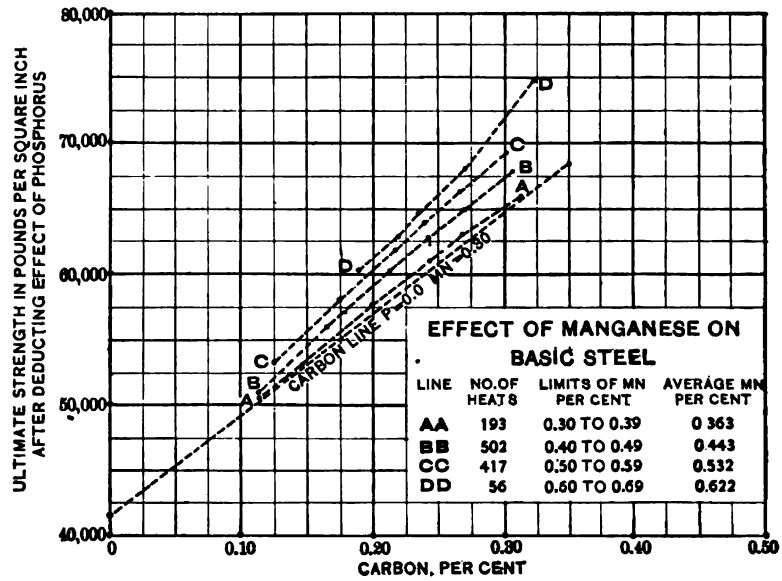
the content of carbon increases, and it is possible to estimate the effect by noting at what point a prolongation of these lines will cut a given ordinate, say the ordinate of 0.60 per cent. of carbon. It is estimated that the line AA will cut this ordinate

FIG. 11.



EFFECT OF MANGANESE ON ACID STEEL.

FIG. 12.



EFFECT OF MANGANESE ON BASIC STEEL.

at 103,000 lb. and it is known that, according to the formula for acid steel, the strength of steel of 0.60 per cent. of carbon and 0.40 per cent. of manganese would be 100,000 pounds. There is therefore an increase in strength of 3,000 lb. due to the amount of manganese in excess of 0.40 per cent. In Table XIX it will be seen that the average manganese of the line AA is about 0.46 per cent., so that the strengthening effect of manganese in a steel of 0.60 per cent. of carbon, as shown by this one line, is 500 lb. for each 0.01 per cent. In a similar way the line BB gives a value of 429 lb., and the line CC, 454 pounds.

The variation in values does not represent any law, but arises from determinative errors and the results should therefore be averaged. The line AA represents 296 heats; BB, 300 heats; and CC, 36 heats, so that the true average is 464 lb. for the influence of 0.01 per cent. of manganese upon a steel of 0.60 per cent. of carbon. In Table VII the corresponding figure was 480 pounds.

The results from the basic steel are plotted in Fig. 12, the base being 0.30 per cent. of manganese instead of 0.40 per cent. as in acid metal. It will be evident that the same law holds good that the effect of manganese increases with higher carbons. An analysis according to the same method used in the acid steels gave a value for 0.01 per cent. of manganese of 200 lb. for a steel of 0.35 per cent. of carbon, when Table XIII calls for 234 pounds. In both the acid and the basic steels the agreement is all that could be expected, so that it would seem that in the steels under consideration the manganese has a value varying with the amount of carbon present.

Conclusions.

Carbon.—In acid steel each 0.01 per cent. of carbon strengthens steel by 1,000 lb. per sq. in. when the carbon is determined by combustion. The strengthening effect is 1,140 lb. for each 0.01 per cent. as determined by color, owing to the fact that the color-test does not determine all the carbon present.

In basic steel each 0.01 per cent. of carbon strengthens steel by 770 lb. per sq. in. when the carbon is determined by combustion. The strengthening effect is 820 lb. for each 0.01 per cent. as determined by color.

Phosphorus.—Each 0.01 per cent. of phosphorus strengthens steel by 1,000 lb. per sq. in.

Manganese.—Each 0.01 per cent. of manganese has a strengthening effect upon steel, and the effect is greater as the content of carbon increases. Below a certain content of manganese the effect is complicated by some disturbing condition, probably iron oxide, so that a decrease in manganese in very low-carbon steels is accompanied by an increase in strength. In acid steel each increase of 0.01 per cent. of manganese above 0.4 per cent. raises the strength of acid steel an amount varying from 80 lb. in a metal containing 0.1 per cent. of carbon to 400 lb. in a metal containing 0.4 per cent. of carbon. In basic steel each increase above 0.3 per cent. raises the strength an amount varying from 130 lb. in a metal containing 0.1 per cent. of carbon, to 250 lb. in a metal containing 0.4 per cent. of carbon.

Sulphur.—The effect of sulphur on the strength of acid and of basic steel is very small.

Formulæ.—From the foregoing results, the following formulæ may be written, in which C = 0.01 per cent. of carbon, P = 0.01 per cent. of phosphorus, Mn = 0.01 per cent. of manganese, R = a variable to allow for heat-treatment, and the answer is the ultimate strength in pounds per square inch. The coefficient of manganese in acid steel, called x, is the value given in Table VII, and applies only to contents above 0.4 per cent. The value of manganese in basic steel, called y, is the value given in Table XIII, and applies to contents above 0.3 per cent.

Formula for acid steel, carbon by combustion:

$$40{,}000 + 1{,}000\,C + 1{,}000\,P + x\,Mn + R = \text{Ultimate Strength.}$$

Formula for acid steel, carbon by color:

$$39{,}800 + 1{,}140\,C + 1{,}000\,P + x\,Mn + R = \text{Ultimate Strength.}$$

Formula for basic steel, carbon by combustion:

$$41{,}500 + 770\,C + 1{,}000\,P + y\,Mn + R = \text{Ultimate Strength.}$$

Formula for basic steel, carbon by color:

$$42{,}000 + 820\,C + 1{,}000\,P + y\,Mn + R = \text{Ultimate Strength.}$$

I am indebted to my brother, J. W. Campbell, of the Pennsylvania Steel Co., for collecting the data and assisting in the arithmetical work involved in this paper.

The Decomposition and Formation of Zinc Sulphate by Heating and Roasting.

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PART I. DECOMPOSITION OF ZINC SULPHATE BY HEATING IN AIR.

I. *Introduction.*

WITH the exception of lead sulphate, all common metallic sulphates are completely decomposed upon heating into metallic oxide, sulphur trioxide, sulphur dioxide and oxygen. Some give up their trioxide readily at a low temperature, others

require considerable heat and much time to be completely freed from sulphur. Kerl[1] in 1881 arranged the principal metallic sulphates as they are decomposed by a rising temperature in the following order: Silver, iron, copper, zinc, nickel, cobalt, manganese and lead; lead sulphate being decomposed only slightly at a white heat. The researches of Bradford,[2] published in 1903, show, however, that ferrous sulphate is decomposed at 590° C., cupric sulphate at 653° C. and argentic sulphate at 1,095° C. By the present investigation it will be seen that zinc sulphate is decomposed at 739° C. As far then as our present knowledge goes, the order in which the decomposition of the leading metallic sulphates takes place is the one given in Table I.

TABLE I.—*Temperatures of Decomposition of Various Metallic Sulphates.*

Fe	Cu	Zn	Ag_2	Ni	Co	Pb	SO_4
590	653	739	1,095	n. d.	n. d.	n. d.	°C.

Very little work has been done to determine the temperature at which zinc sulphate is decomposed, the rapidity with which the decomposition takes place, and the products that are formed. Schmieder[3] approximated the temperature of complete decomposition of zinc sulphate in a hand-reverberatory roasting-furnace treating blende, by placing Seger cones near the fire-bridge and noting the one that melted when basic sulphate was converted into oxide. He estimated the temperature to be about 1,100° C.; that is, the hearth near the fire-bridge has to reach about 1,100° C., if the blende is to be satisfactorily dead-roasted. As to the time required for decomposing zinc sulphate, it is known in a general way that dead-roasting blende is a lengthy operation on account of the difficulty of decomposing the last of the sulphate. Eichhorn,[4] in connection with roasting blende in a muffle furnace, calls attention to the fact that it is not so much temperature as time that is required to

[1] *Grundriss der Metallhuettenkunde*, Leipsic, 1881, p. 70.
[2] *Trans.*, xxxiii., 50.
[3] *Œsterreichisches Jahrbuch*, 1889, p. 393.
[4] *Berg- und Huettenmœnnische Zeitung*, 1889, p. 113.

obtain satisfactory results. In regard to the products formed, Vogel,[5] as early as 1814, brought normal zinc sulphate to a red heat and obtained a product insoluble in water which contained sulphur. He called it basic sulphate. Plattner[6] says that in roasting blende there is formed, beside the normal sulphate, a basic sulphate of definite chemical composition, the tetrabasic sulphate 3 ZnO, $ZnSO_4$, and that while the former is readily decomposed upon raising the temperature above the normal for roasting, into ZnO, SO_3, SO_2 and O, the basic salt requires a white heat for the complete elimination of sulphur. Plattner is the only investigator who gives a basic salt formed by heating normal sulphate a definite formula. The present investigation makes it probable that, in decomposing zinc sulphate by heat, the sulphur trioxide passes off gradually and that no definite basic compound is formed.

A series of basic salts crystallizing with various amounts of water has been isolated from solutions, but they have no bearing on the work in hand.

Some of the preliminary work in this investigation was done by Messrs. W. A. Dorey and R. P. Roberts in 1900. It was carried further in 1901–02 by Mr. A. C. Dart, Jr., who tried to prove or disprove, by heating zinc sulphate in a current of air and of carbon dioxide, the correctness of the generally accepted statement, that heating a metallic sulphate with limited access of air causes the SO_3 to split to a large extent into SO_2 and O, while with free access of air the volatile compound driven off is mainly SO_3. His results are given in Table II.

TABLE II.—*Results of Heating Zinc Sulphate With and Without Free Access of Air.*

Heated in	Temperature.	Total Sulphur Eliminated.	Sulphur Eliminated as SO_2.	Sulphur Eliminated as SO_3.	Ratio: $\frac{S \text{ as } SO_2}{S \text{ as } SO_3}$
	Degrees Centigrade.	Per Cent.	Per Cent.	Per Cent.	
Air	578	1.63	0.14	1.49	9.06
Air	584	1.79	0.14	1.65	8.26
Carbon dioxide	599	0.50	0.18	0.32	55.25
Carbon dioxide	602	0.50	0.16	0.34	47.88

[5] *Journal fuer Chemie und Physik* (edited by J. S. C. Schweigger), 1814, xi., p. 416.

[6] *Die Metallurgischen Roestprocesse*, Freiberg, 1856, pp. 87, 142.

In the experiments, the results of which are given in Table II, the temperature of the furnace was brought slowly to the point at which the first acid was given off, then raised about 10° C. and maintained constant for several hours. While only a very small proportion of the total sulphur was driven off in the tests, they show that in heating zinc sulphate the tendency of the salt to split into ZnO, SO_3 and O is greater when oxygen is absent than when it has free access. Schnabel[7] quotes Schlapp as having found, while decomposing zinc sulphate on a large scale by heating, that of the total SO_3 present, 30 per cent. escaped as such, while the rest passed off as SO_2 and O. Investigations along this line were not further pursued, as the researches of Knietsch[8] on the formation and decomposition of sulphur trioxide by heat in the presence and absence of catalyzing agents had about covered the ground.

The present investigation was taken up in 1903. A grant by the American Academy of Arts and Sciences from the income of the C. M. Warren fund enabled me to avail myself of the assistance of Mr. R. B. Yerxa in carrying on the work.

II. *Zinc Sulphate Used.*

The zinc sulphate which formed the raw material for the experiments was the chemically pure normal salt, $ZnSO_4 + 7\,H_2O$. It was analyzed for zinc by the standard sodium carbonate method, and for sulphur by precipitation with barium chloride, the zinc having been first removed to prevent any of it from being carried down by the barium sulphate. The water was found by the difference. The results of the analysis are given in Table III.

TABLE III.—*Composition of Zinc Sulphate.*

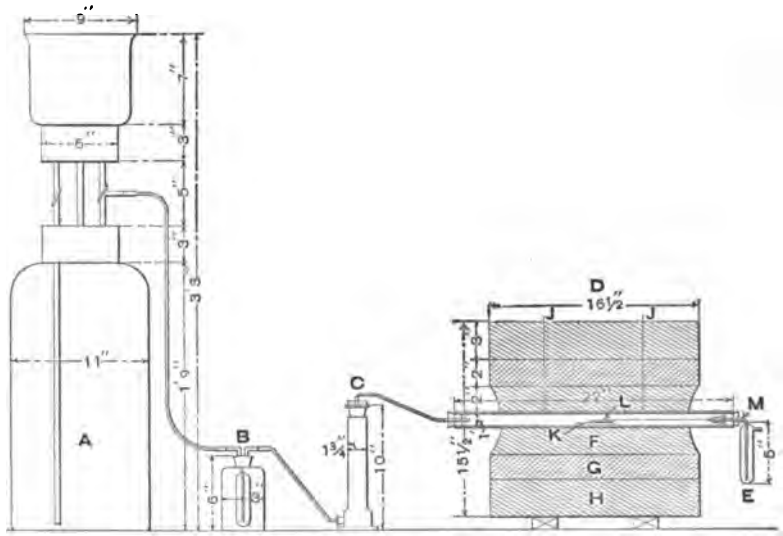
	By Analysis.	By Calculation.
	Per Cent.	Per Cent.
SO_3	28.78	27.84
ZnO	29.27	28.32
	By difference.	
H_2O	41.95	43.84
Ratio : SO_3 : ZnO	0.9833	0.9828

[7] *Handbuch der Metallhuettenkunde*, Springer, Berlin, 1901, 2d ed., vol. ii., p. 46.

[8] *Berichte der Deutschen Chemischen Gesellschaft*, 1901, xxxiv., 4,069; transl. by Ruhoff, *Mineral Industry*, 1901, x., 605.

The results in Table III show that the analysis gave about 2 per cent. less water than the quantity required by the formula. This difference was accounted for by the fact that the salt had been kept in a closet the temperature of which had sometimes reached 30° C. Upon close examination the crystals appeared dull, as if they had weathered. The close agreement of the two ratios of $SO_3 : ZnO$, proves that the sulphate was the normal salt.

FIG. 1.



ELECTRIC TUBE-FURNACE WITH GAS-TRAIN.

A, Gasometer; B, Allihn wash-bottle with permanganate; C, Drying tower; D, Tube-furnace; E, Railroad-tube with indicator-solution; F, Calcined magnesia; G, Asbestos-magnesia; H, Fluted asbestos steam-pipe covering; J, Porcelain tube; K, Platinum boat; L, Thermo-couple; M, Rubber stopper.

III. *Furnaces and Pyrometers.*

Two electric furnaces and one gas-furnace served for the heat-work. The electric furnaces were of the resistance type first suggested and used by Prof. C. L. Norton[9] in 1894, and since then commonly employed for laboratory heating-purposes with temperatures up to about 1,400° C. Fig. 1 represents a sketch of the tube-furnace, D, with gas-train, A, B, C. A porcelain tube, J, J, 22 in. long and 1 in. internal diameter, glazed

[9] Fay and Badlam, *Technology Quarterly*, 1900, xiii., 304; Norton, *Iron Age*, Jan. 3, 1901, p. 22.

inside only, was wound in the center for a distance of about 8 in. with 0.5-mm. platinum wire. The platinum coil, of 30 turns about 0.25 in. apart, was coated with a thin mortar of chemically pure magnesia; the tube then covered, excepting 5 in. at the ends, with a thick mortar of calcined magnesia, F; this surrounded with ordinary asbestos-magnesia covering, G; and the whole enclosed with fluted asbestos steam-pipe covering, H. The sides of the furnace were firmly held together with brass ribbons in common use in covering steam-pipes, and the ends plastered over with a mixture of magnesia and water-glass to prevent the dry part of the lining from falling out. The platinum coils were coated with chemically pure magnesia to prevent them from coming in contact with the impurities contained in the ordinary magnesia, as these often form clinkers which corrode the platinum wire. By leaving the ends of the porcelain tube uncovered for 5 in., they remained sufficiently cool, when the furnace was brought to the highest temperature, to admit the use of rubber stoppers. The wires of the thermo-couple were firmly connected with the rubber stopper, M, at the delivery-end of the tube by cutting a longitudinal slit into opposite sides, inserting a wire, and binding the whole with adhesive tape. By thus fastening the wires to the stopper, the couple, L, when once adjusted to be in contact with the substance in boat, K, to be heated, would be again in correct position when the tube was closed up after the stopper had been removed for some reason or other. The wires reaching into the tube were insulated from one another, yet, at the same time, firmly held together by winding them over and under, with asbestos string.[10]

The gas-train consists of a gasometer, A, of ordinary type used for forcing air or any other gas through the apparatus. It is followed by an Allihn wash-bottle, B, filled with a strong solution of potassium permanganate, which oxidizes all traces of organic matter or sulphurous acid from the air. The drying tower, C, filled with pieces of calcium chloride, removes any moisture present, so that only thoroughly purified and dried air can enter the porcelain tube of the furnace. At the opposite end is a railroad tube, E, to hold the indicator solution. It was found necessary to have two indicators. One was a very dilute

[10] Le Chatelier-Boudouard-Burgess, *High Temperature Measurements*, Wiley, New York, 1901, p. 113.

solution of iodine in potassium iodide and of starch, to show the presence of sulphurous acid by the fading of the blue color. As the blue color is also destroyed by organic matter, another indicator had to be provided for carbon-reduction experiments. At the suggestion of Prof. A. H. Gill, a colorless solution of potassium iodate and starch was used which was changed to blue by sulphurous acid. Carbon burnt in the tube had no effect upon it.

The tube-furnace was adopted after experimenting for some time with a similar electric furnace, 16 in. long by 10 in. in diameter, the chamber of which consisted of a cylindrical clay retort, 6 in. long by 2.5 in. in diameter, the mouth being flush with the covering to admit access with tools. The furnace had to be given up for experiments requiring hermetic sealing, as it was found impossible to cement a plug into the mouth of the cylinder which would be air-tight during the experiment and permit of being easily removed at the close of a test. The retort-furnace was employed, however, in conjunction with the tube-furnace for dehydrating and for heating larger quantities of zinc sulphate than the tube could conveniently hold under conditions not requiring the furnace to be air-tight. A gas-muffle furnace, made by the American Gas Furnace Co., New York, N. Y., with muffle 12 in. long by 6.75 in. wide by 3.75 in. high, was used for heating large quantities of zinc sulphate and for roasting blende.

The temperatures were measured with Le Chatelier thermo-electric couples. With the tube- and retort-furnaces a Chauvin-Arnoux galvanometer was employed; with the gas muffle-furnace the common form of Deprez-d'Arsonval. The couples were calibrated against the boiling-points of water, naphthalene and sulphur, and the melting-points of aluminum and gold.

IV. *Heating Zinc Sulphate—Dehydration.*

Manuals of chemistry teach that hydrous zinc sulphate loses six of its seven molecules of water when it is heated to 100° C.; that, at a higher temperature, the last molecule is driven off; and that it is difficult to remove this without, at the same time, starting the decomposition of the anhydrous salt. As it was important in the proposed work to start with anhydrous salt,

tests were made to find the temperature at which zinc sulphate was completely dehydrated. Weighed samples of the crystalline salt were partly dehydrated on a steam-table and then transferred to the tube-furnace. The temperature of the steam-table was about 109° C. which caused an average loss in weight of 36.29 per cent. In the tube-furnace the temperature was first brought to, and maintained at, 238° C. to constant weight; it was next raised to 324° C. and again held until there was no further diminution in weight. Increasing the heat did not cause any additional loss until the sulphate began to be decomposed. The temperature of approximately complete dehydration was, however, lower. Further tests at 240°, 260° and 280° C. showed that above 280° C. there was no additional loss. Working within the limits of 260° and 280° C. it was found that 263° C. was the lowest temperature at which the last water (excepting 0.24 per cent.) was given off. With samples weighing 2 g. each, it took from 30 to 40 hours to finish a test. The average loss of water in the runs made at 263° C., or slightly above, was 41.71 per cent., while the analysis of the salt calls for 41.95 per cent. The last 0.24 per cent. water could be set free only at a temperature at which the decomposition of the sulphate began. The losses in weight from 100° to 263° C. showed that the quantity of water driven off rose with the temperature, as did the rapidity with which it passed off.

V. *Heating Zinc Sulphate—Decomposition.*

(*a.*) *Beginning of Decomposition.*—Five separate experiments were carried through to determine the beginning of decomposition of anhydrous zinc sulphate. The mode of operating was as follows: A sample of the hydrous salt weighing about 3 g. was partially dehydrated on a steam-table, transferred to the tube-furnace, heated to constant weight, set aside to free the cool end of the delivery-tube completely from condensed moisture, and then replaced in the tube. In the preliminary trials it has been found that this condensed moisture absorbed the first traces of acid set free and vitiated the results. With the sample again in place and the tube closed, the air-current was started and the furnace heated, rapidly to 500° C. and then slowly. Galvanometric readings were taken as a rule every minute, often, however, at shorter intervals. When, in the

slow heating, the temperature was raised 5° C. a minute, the first fading of the indicator occurred at 550° C.; when more slowly, it took place at a lower temperature. The lowest point recorded for the beginning of decomposition on a slowly rising temperature was 532° C. Upon lowering the temperature from 532° to 525° C. and allowing it to remain constant, no fading of the indicator could be detected. When it was increased to 528° C. and held there, the indicator began to fade after about 10 minutes. The decomposition of anhydrous zinc sulphate begins at 532° C. with a rising, and at 528° C. with a constant, temperature. The difference of 4° C. finds its explanation in the time required for the extremely small quantity of acid set free to travel from the boat, holding the charge, to the indicator tube. The evolution of sulphurous gas at 528° C. lasted for from 1 to 2 hours, according to the weight and the thickness of the charge. In two tests in which the salt was weighed after the sulphurous gas had ceased to be evolved, the loss in weight was only 0.04 and 0.05 per cent., based on that of the anhydrous salt, which was about 2 g. It ought to be mentioned that, after one test was finished and before another was begun, the temperature was lowered well beneath the decomposition-point; also, the furnace was swept clean from any remaining free acid by a current of air.

(*b.*) *Rapidity of Decomposition.*—In attempting to determine the rapidity of decomposition it was necessary to feel one's way before any systematic work could be undertaken. A sample corresponding to 1.7535 g. of anhydrous zinc sulphate was first heated to 528° C. and kept at this temperature for 1.5 hours, when acid ceased to come off. No further decomposition took place with increase of temperature until 575° C. was reached. It took 3.5 hours of heating to separate the acid. Raising the temperature again slowly, the indicator showed the presence of acid at 610° C., and 52.5 hours passed before acid ceased to come off. As it required at the end of the 52.5 hours one whole hour's heating for the indicator to show signs of fading, the temperature was raised more quickly, 10° C. per hour, until 681° C. was reached and maintained at this stage for 4 hours. A further rise of 9° C. brought the charge to 690° C., at which temperature it was again held for 4 hours. The result of the 76 hours' heating was a loss of only 7.5 per cent. of SO_3 out of

a possibility of 49.6 per cent. This decrease in weight was checked by a determination of sulphur in the residue, which was found to be 16.8 per cent. The 7.5 per cent. of SO_3 correspond to 3 per cent. of sulphur, and the anhydrous zinc sulphate contained 19.84 per cent. of sulphur. In all these tests the evolution of sulphurous acid was comparatively rapid when a decomposition-stage was reached, and then slowed down until it finally ceased altogether.

The preliminary tests having shown that with low temperatures the evolution of sulphurous acid gas was excessively slow, the subsequent tests were carried on at higher temperatures and with shorter intervals between temperature-raisings; further, the residue was weighed at the end of each heat, and thus the percentage-loss by volatilization established. Five runs were made to study the general behavior of the salt unto complete decomposition, the rate of decomposition at certain temperatures, and the effect of the thickness of the charge. Table IV gives the losses in weight obtained in raising the temperatures from 528° to 836° C. in the stated time-intervals. The loss in weight is very small at first; it increases, however, with the temperature. Six hours' heating at 755° C. causes a loss in weight (8.62 per cent.) far greater than 72 hours' heating at or slightly above 610° C. in the preceding run (7.5 per cent.). After having been kept for 24 hours at 755° C., the loss in weight has fallen to 1.81 per cent. (the lowest decomposition temperature is 739° C., as will be shown later on), while the total loss amounts to only 22.23 per cent. As this progress was too slow, the temperatures were raised more quickly.

Table V gives the results of the tests made at 836° C., the highest temperature reached in Table IV, varying the thickness of the charge by spreading the same weight of ore over a 3-inch and a 6-inch boat, which gave layers 0.25 in. and $\frac{1}{16}$ in. in thickness. The 3-in. boat was heated in the tube-furnace, the 6-in. boat, in the retort-furnace. The results are plotted in curves **A** and **B** of Fig. 2.

Table VI gives the results of the tests made at 803° and 770° C., the salt having been spread in a 3-in. boat to a thickness of $\frac{1}{64}$ in. The results are plotted in curves **C** and **D** of Fig. 2.

TABLE IV.—*Losses in Weight by Heating Zinc Sulphate.*

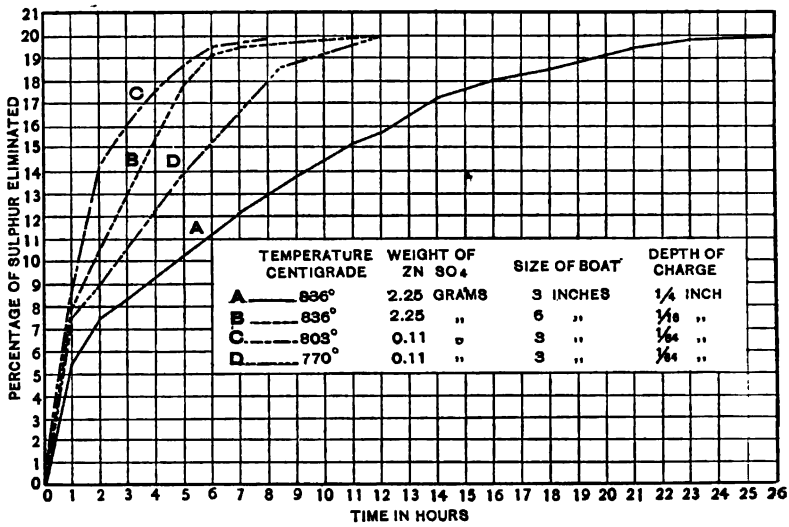
[Charge: 1.9562 g. of zinc sulphate in a 3-in. boat.]

Time of Each Heat.	Temperature.	Loss of SO_3 in Each Heat.	Total Loss of SO_3.
Hours.	Degrees Centigrade.	Per Cent.	Per Cent.
1	528	0.04	0.04
4	618	0.46	0.50
6	636	2.19	2.69
6	755	8.62	11.31
6	755	6.95	18.26
6	755	2.16	20.42
6	755	1.81	22.23
6	760	2.27	24.50
3	768	1.67	26.37
3	785	2.72	28.11
3	792	4.83	32.94
3	803	5.50	38.44
3	815	5.08	43.52
3	826	4.71	48.23
3	836	1.27	49.50

TABLE V.—*Losses in Weight of Zinc Sulphate Heated to* 836° *C.*

Charge: 2.255 g. $ZnSO_4$ in a 3-in. Boat.			Charge: 2.259 g. $ZnSO_4$ in a 6-in. Boat.		
Time.	Loss SO_3.	Loss S (Calculated).	Time.	Loss SO_3.	Loss S (Calculated).
Hours.	Per Cent.	Per Cent.	Hours.	Per Cent.	Per Cent.
1	13.60	5.44	1	19.78	7.91
2	18.57	7.43	2	26.37	10.55
3	20.68	8.27	3	32.70	13.08
4	23.14	9.26	4	38.98	15.59
5	25.79	10.28	5	44.58	17.83
6	27.64	11.07	6	47.74	19.10
7	30.04	12.02	7	48.64	19.46
8	32.09	12.84	8	49.11	19.64
9	34.06	13.62	9	49.50	19.80
10	35.90	14.36	10	49.61	19.84
11	37.93	15.17	11	49.79	19.91
12	39.72	15.88	12	49.96	19.98
13	41.45	16.58			
14	42.83	17.13			
15	44.16	17.66			
16	45.10	18.04			
17	46.00	14.40			
18	46.59	18.64			
19	46.98	18.79			
20	47.81	19.12			
21	48.59	19.44			
22	48.86	19.54			
23	49.33	19.73			
24	49.62	19.85			
26	49.77	19.90			

FIG. 2.



RAPIDITY OF DECOMPOSITION OF ZINC SULPHATE.

TABLE VI.—*Losses in Weight of Zinc Sulphate Heated to 803° C. and 770° C.*

Temperature, 803° C.			Temperature, 770° C.		
Charge : 0.109 g. of $ZnSO_4$ in a 3-in. Boat.			Charge : 0.1179 g. of $ZnSO_4$ in a 3-in. Boat.		
Time.	Loss SO_3.	Loss S (Calculated).	Time.	Loss SO_3.	Loss S (Calculated).
Hours.	Per Cent.	Per Cent.	Hours.	Per Cent.	Per Cent.
1.....	20.18	8.07	1.....	18.82	7.53
2.....	35.59	14.24	2.....	22.30	8.92
3.....	40.00	16.00	3.....	26.29	10.51
4.....	43.95	17.58	4.....	30.45	12.18
5.....	46.61	18.64	5.....	34.61	13.84
6.....	48.63	19.45	8.5.....	46.40	18.56
7.....	49.09	19.64	11.5.....	49.37	19.57
8.....	49.55	19.82	12.....	49.96	19.98

Comparing curves **A** and **B** in Fig. 2, the importance of the thickness of charge upon the elimination of sulphur is brought out clearly. The charge and the temperature are the same in both cases. With a charge 0.25 in. thick (curve **A**), it takes 26 hours for complete desulphatization, while 12 hours (curve **B**) are sufficient to do the same work, if the charge has a depth of $\frac{1}{16}$ in. The loss in sulphur during the first hour's heating

is rapid in both instances, but less so with the thicker charge (5.44 per cent.) than with the thinner one (7.91 per cent.); it then decreases gradually, again more rapidly with the thin than the thick charge. At the end of 6 hours the thick charge has lost 11.07 per cent. of sulphur, while with the thin charge 19.1 per cent. or nearly all the sulphur has been eliminated, yet it requires 6 hours more to drive off the remaining traces. When, after 12 hours, the thin charge has been freed from sulphur, the thick charge has lost only 15.88 per cent., and it requires 14 hours more heating to remove the remaining 4 per cent. of sulphur. Curves **C** and **D**, representing the heating of a small amount of salt at 803° and 770° C., are both steep; they show again the advantage of working with a thin charge and bring out clearly the effect of difference of temperature. With 803° C., the decomposition takes 8 hours, with 770° C., 12 hours. In the first hour the loss in sulphur is nearly the same, 8.07 per cent. (803° C.) and 7.53 per cent. (770° C.); the curves then diverge, the high-temperature curve rising quite rapidly as against the one of lower temperature. Comparing curve **B** (836° C., thickness of charge $\frac{1}{16}$ in.) and curve **D** (770° C., thickness of charge $\frac{1}{64}$ in.), it is seen that both charges are completely decomposed after 12 hours' heating; the elimination of sulphur at the higher temperature is more rapid in the first 6-hours' than at the lower temperature. It requires, however, 6 additional hours to remove the last 0.88 per cent. of sulphur at 836° C. with a $\frac{1}{16}$-in. layer, while only 2 hours are necessary to do the same work at 770° C. with a $\frac{1}{64}$-in. layer.

It ought to be added that in the runs represented by curves **A, B, C** and **D**, the heating was carried on 1 hour longer than shown in the tables and in Fig. 2, in order to have proof positive that no further loss took place.

(*c.*) *End of Decomposition.*—In the experiments recorded in Tables V and VI and represented graphically in Fig. 2, anhydrous zinc sulphate was completely decomposed at 770°, 803° and 836° C. It was of interest to find whether 770° C. was the lowest temperature of complete decomposition or not. For this purpose the experiments recorded in Table VII were carried out. The charge of 0.1169 g. of dehydrated zinc sulphate in a 3-in. boat, was brought quickly to 763° C. and maintained for

4 hours at this temperature. This was done to hasten the elimination of sulphur; weighings were made hourly to note the progress. The temperature was then lowered a few degrees at a time at half-hour intervals until no further loss in weight occurred. This took place at 730° C. The temperature was now raised to 735° C. and held there for 5.5 hours, half-hourly weights being taken.

TABLE VII.—*Losses in Weight of Zinc Sulphate Heated Rapidly to 763° C.*

[Charge: 0.1169 g. of zinc sulphate in a 3-in. boat.]

Time of Each Heating	Temperature.	Loss in Weight.	Loss of SO_3.
Hours.	Degrees Centigrade.	Grams.	Per Cent.
4	763	0.0310	26.52
0.5	752	0.0020	1.71
1	748	0.0032	2.74
0.5	739	0.0016	1.37
0.5	730	0.0000	0.00
5	735	0.0132	11.29
0.5	735	0.0000	0.00
1	777	0.0033	2.82
1	737	0.0013	1.11
0.5	737	0.0000	0.00
0.5	739	0.0014	1.20
0.5	739	0.0004	0.34
0.5	739	0.0000	0.00
0.5	749	0.0000	0.00
1.5	754	0.0000	0.00
0.5	842	0.0000	0.00

As there still remained in the residue 5.47 per cent. of sulphur trioxide, the temperature was brought to and maintained at 777° C. for 1 hour to drive off the bulk of the 5.47 per cent. Table VII shows that 2.82 per cent. was thus removed. Lowering the heat to 737° C. and holding it there for 1 hour caused a loss of 1.11 per cent., but an additional half hour caused no further loss, although there still remained 1.54 per cent. of acid. Raising the temperature 2° to 739° C. caused in half an hour a loss of 1.2 per cent., and of 0.34 per cent. in the next half hour; a third half hour at 739° C. caused no further loss in weight, nor did heating to 749°, 754° and 842° C. The decomposition had been completed at 739° C.; the total loss in weight was 49.1 per cent.; the time 14 hours. The time required for decomposing the salt completely at 739° C. was not ascertained, but, judging by the runs at 803° and 770° C., it would have been of long duration.

(*d.*) *Retardation in Heating-Curve.*—All the experiments so far carried out point to the conclusion that stable anhydrous basic zinc sulphates do not exist; the sulphur trioxide of the normal sulphate passes off gradually without any decided interruption, as the temperature is raised from beginning to complete decomposition. In order to verify this conclusion, the method employed by Bradford[11] in finding the temperatures of decomposition of ferrous, cupric and argentic sulphate was applied to zinc sulphate. It consists in heating the salt slowly in contact with a thermo-couple to and above decomposition, and recording the rise of temperature. Upon dissociation of the substance there will be caused a break in the heating-curve; it will be a retardation if heat is absorbed, an acceleration if set free. Zinc oxide combining with sulphur trioxide forms an exothermic reaction,

$$ZnO + SO_3 = ZnSO_4 + 52.9 \text{ Kg. cal.};^{12}$$

hence, upon being decomposed, heat is absorbed, and the heating-curve will show a decided retardation, if all the sulphur trioxide is given off readily at a certain temperature, or if the normal anhydrous salt passes through clearly defined stops of basic salts while it is being converted into oxide.

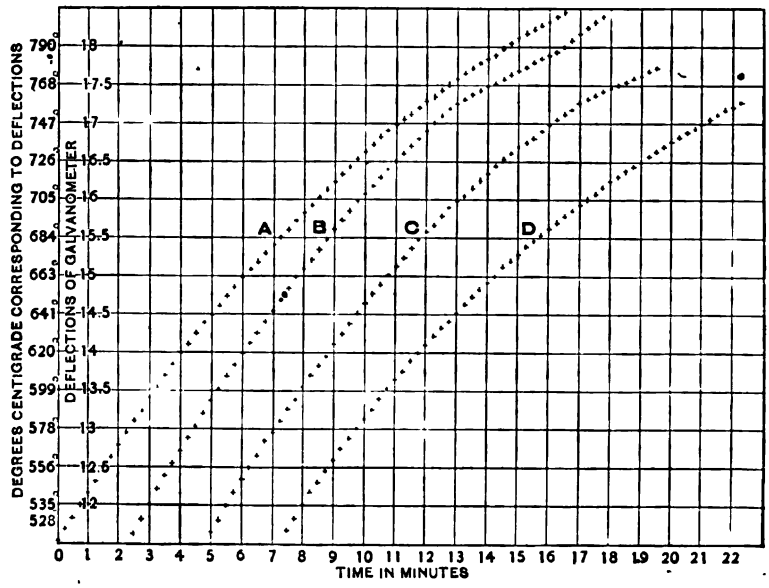
The retort-furnace was used in the tests. It was placed on end and the retort partly filled with magnesia; a porcelain crucible charged with 5 g. of anhydrous zinc sulphate was embedded up to the rim in the magnesia. The junction of the thermo-couple was placed in the salt so as nearly to touch the bottom of the crucible. The current of 110 volts was turned on with the resistance-coils cut out. This gave at first a rise in temperature of 20° C. per minute, but it decreased, of course, as the temperature of the crucible rose. The galvanometer was read every 15 seconds; the readings were begun at little below 528° C. and continued to above 740° C. Four separate runs were made, three with the salt in the crucible, one with the empty crucible to serve as a blank test. In the plot, shown in Fig. 3, the galvanometer deflections and corresponding temperatures in degrees centigrade scale are plotted on the ordinate and the time in minutes on the abscissa. Curve A represents the blank

[11] *Trans.*, xxxiii., 50.

[12] Berthelot, *Thermo-Chimie*, Paris, 1897, vol. ii., pp. 91, 305, 309.

test, curves B, C and D the runs made with the charged crucible. Curve A shows slight retardations near 747° and 790° C., which are due to irregularities of current, caused probably by current being turned on or shut off in other parts of the Institute's laboratories. The apparent retardations in curve B at 760° C. and in curve C at 735° and 765° C. must be due to the same cause as that in curve A, since in curve D no retardation whatever is perceptible. These results therefore substantiate the conclusions formed from the preceding work.

FIG. 3.



HEATING-CURVE OF ZINC SULPHATE.

(*e.*) *Formation of Basic Salts.*—When anhydrous zinc sulphate, partly decomposed by heating, is treated with hot water, the undecomposed salt is quickly hydrated and dissolved, leaving behind basic salt, which dissolves with the greatest difficulty in water with continued boiling, small samples requiring days of treatment. The following experiments were planned to see how much basic sulphate was formed as the decomposition of the normal sulphate and the elimination of sulphur progressed. The heating was carried on in the gas muffle-furnace. A charge of 80 g. of anhydrous normal zinc sulphate was placed in an oblong clay roasting-dish, 4.25 by 10.25 in. and 0.75 in. deep, with walls

0.25 in. thick. The temperatures were taken at intervals of a few minutes by placing the thermo-couple in contact with the salt. As the top of this was found to be 10° C. lower than the bottom, the couple was pushed down toward the bottom of the tray. The charge, 0.5 in. in thickness, was rabbled (with a few exceptions to be noted later) at intervals of a few minutes, and samples weighing from 10 to 14 g. were taken every hour. After a sample had been obtained, the salt was pushed together so as to have the same depth as the original charge. Three heatings were made: at 790° C., or 50° C. above the lowest temperature of complete decomposition of zinc sulphate; at 740° C., the temperature of decomposition; and at 690° C., 50° C. below the final temperature. It was necessary to make on each sample three analytical determinations,—the total zinc, the zinc soluble in hot water, and the sulphur insoluble in hot water.

The total sulphur eliminated was calculated by deducting from the sulphur in charge, the insoluble sulphur and the sulphur in combination with the soluble zinc sulphate. Zinc was determined by titrating with potassium ferrocyanide and the residual sulphur by dissolving the basic sulphate in the smallest possible quantity of hydrochloric acid and then precipitating with barium chloride. The total zinc-content of each sample was ascertained in order that the analytical results might be reduced to a common basis, namely—to the basis of 40.52 per

TABLE VIII.—*Basic Zinc Sulphate Formed by Heating Zinc Sulphate.*

[Charge: 80 g. of $ZnSO_4$; temperature, 790° C.; time, 6 hours.]

	Actual Analyses.					
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Zinc, total	43.44	48.53	48.97	49.49	52.05	54.49
Zinc as normal sulphate	36.28	26.40	25.73	24.62	22.02	19.82
Sulphur as basic sulphate	0.80	2.65	2.66	2.77	2.85	2.95
Analyses Corrected to Basis of 40.52 Per Cent. of Zinc.						
Zinc as normal sulphate	33.84	22.01	21.29	20.16	17.38	14.72
Sulphur as basic sulphate	0.747	2.21	2.20	2.27	2.25	2.14
Sulphur eliminated (calculated)	2.69	7.05	7.42	7.91	9.32	10.76

Average ratio: $\frac{\text{Sulphur eliminated}}{\text{Sulphur as basic sulphate}} = 3.82.$

cent. of zinc contained in the normal anhydrous sulphate. The results of the actual and corrected analyses, and of the calculated corrected figures for eliminations of sulphur, are given in Tables VIII (790° C.), IX (740° C.) and X (690° C.); the corrected figures are plotted in Fig. 4.

TABLE IX.—*Basic Zinc Sulphate Formed by Heating Zinc Sulphate.*

[Charge: 80 g. of $ZnSO_4$; temperature, 740° C.; time, 6 hours.]

	Actual Analyses.					
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Zinc, total	40.82	42.27	43.27	44.83	45.74	46.55
Zinc as normal sulphate	38.90	36.88	35.69	33.29	31.09	29.20
Sulphur as basic sulphate	0.189	0.548	0.758	1.10	1.39	1.75
Analyses Corrected to Basis of 40.52 Per Cent. of Zinc.						
Zinc as normal sulphate	38.62	35.35	33.41	30.07	27.54	25.42
Sulphur as basic sulphate	0.188	0.525	0.709	0.997	1.23	1.53
Sulphur eliminated (calculated)	0.762	2.06	2.85	4.23	5.26	6.02

Average Ratio: $\frac{\text{Sulphur eliminated}}{\text{Sulphur as basic sulphate}} = 4.08.$

TABLE X.—*Basic Zinc Sulphate Formed by Heating Zinc Sulphate.*

[Charge: 80 g. of $ZnSO_4$; temperature, 690° C.; time, 6 hours.]

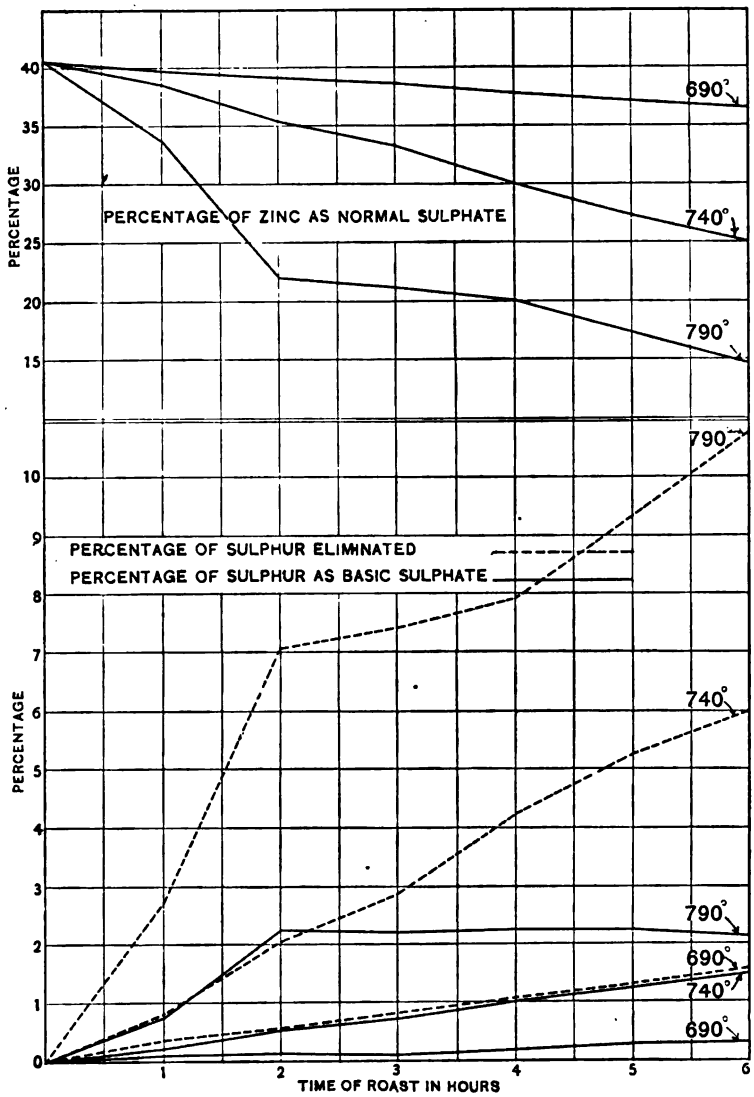
	Actual Analyses.					
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Zinc, total	40.53	40.60	40.63	41.03	41.11	41.37
Zinc as normal sulphate	39.69	39.28	38.84	38.47	37.91	37.40
Sulphur as basic sulphate	0.088	0.112	0.094	0.194	0.297	0.324
Analyses Corrected to Basis of 40.52 Per Cent. of Zinc.						
Zinc as normal sulphate	39.68	39.20	38.72	37.98	37.36	36.72
Sulphur as basic sulphate	0.088	0.112	0.093	0.191	0.293	0.317
Sulphur eliminated (calculated)	0.33	0.55	0.81	1.08	1.29	1.58

Average Ratio: $\frac{\text{Sulphur eliminated}}{\text{Sulphur as basic sulphate}} = 5.4.$

The values for total zinc show readily how far the decomposition of the normal sulphate was carried. Starting with 40.52

per cent. of zinc in the original charge, the percentage has risen at the terminations of the tests, with 790° C. to 54.59,

FIG. 4.



THE FORMATION OF BASIC ZINC SULPHATE.

with 740° C. to 46.55, and with 690° C. to only 41.37. The elimination of sulphur with 790° C. is rapid at first, slackens and then again increases; with 740° and 690° C. it is more

uniform. The total proportion driven off is, of course, greater with the higher than with the lower temperature. A proportionality between the amount of sulphur eliminated and the degree of heat is indicated by the results at the ends of the roasts. The elimination of 10.76 per cent. at 790° C. fell to 6.02 per cent. at 740° C. and to 1.58 per cent. at 690° C.; or in a 6-hour heat a difference of 50° C. in temperature made a difference of 4.5 per cent. in the sulphur elimination.

In the tube-furnace, treating small amounts at a time, it had been noticed that at any given temperature the decomposition was at first rapid and then proceeded more slowly. In the muffle-furnace this was not the case, the difference being no doubt due to the stirring of the charge.

The percentage of sulphur present as basic sulphate increases with the high temperature of 790° C. for the first 3 hours, remains constant up to the fifth hour and then begins to decrease; with the lower temperatures of 740° and 690° C. it shows a steady increase. At the end of the heating, the remaining percentage is higher with 790° than with 740° or 690° C. The average ratio of sulphur eliminated to sulphur as basic sulphate increases as the temperature decreases; it is 3.82 with 790°, 4.08 with 740° and 5.40 with 640° C. The percentage of normal zinc sulphate decreases as the temperature rises. With 790° C. there is a quick fall during the first 2 hours, during the next 2 hours it is slow, and accelerates again the last 2 hours. With 740° and 690° it is more uniform and, of course, greater with 740° than with 690° C.

PART II. DECOMPOSITION OF ZINC SULPHATE BY HEATING WITH CARBON (REDUCING-ROAST).

I. *Introduction.*

If metallic sulphates that are decomposed by heat alone are heated with a carbonaceous reducing agent they can be partly reduced, the sulphur trioxide oxidizing the carbon. This may be expressed by the equation

$$2\,MSO_4 + C = 2\,MO + 2\,SO_2 + CO_2.$$

If the metal has more than one oxide, the higher oxide may be brought to a lower state of oxidation.

$$4\ MSO_4 + 3\ C = 2\ M_2O + 4\ SO_2 + 3\ CO_2.$$

If the metallic oxide is readily reducible, it may be partly brought to the metallic state and, combining with the sulphur, obtained by the reduction of sulphur dioxide, form a metallic sulphide.

$$2\ MSO_4 + C = 2\ MO + 2\ SO_2 + CO_2,$$
$$2\ MO + C = 2\ M + CO_2,$$
$$2\ SO_2 + 2\ C = 2\ S + 2\ CO_2,$$
$$2\ M + 2\ S = 2\ MS.$$

If the temperature is sufficiently high, the metallic sulphate is reduced to sulphide direct.

$$MSO_4 + 2\ C = MS + 2\ CO_2.$$

Reducing-roasts are not frequent now in connection with metallic sulphates; occasionally they are practiced in roasting blende carrying small amounts of galena, as the ready fusibility of galena and of some of its compounds makes it inadvisable to raise the charge to the temperature necessary for driving off all, or at least most, of the sulphur trioxide.

The only temperature measurements made in a reducing-roast are those by Scherr,[13] who found, upon addition of charcoal at 560° C. to partly roasted matte, that the 5 per cent. of soluble cupric sulphate was quickly reduced to 0.2 per cent. The first statement regarding zinc sulphate is that of Guy Lussac,[14] who investigated the behavior of the sulphates of zinc, iron, manganese, magnesium, nickel, lead, copper, silver and quicksilver in a reducing-roast. He found that if zinc sulphate, mixed with an excess of carbon over that required by theory, was brought in a closed vessel to a dark-red heat, there remained a residue of zinc oxide and carbon, while there passed off a gas consisting of 2 volumes of sulphur dioxide and 1 volume of carbon dioxide. If the mixture was brought quickly to a white heat, the residue was zinc sulphide and carbon, while the gas that passed off consisted of carbon monoxide, with a slight admixture of sulphur dioxide and carbon dioxide.

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[13] *School of Mines Quarterly*, 1900, xxi., 66.

[14] *Erdmann's Journal fuer Praktische Chemie*, 1837, xi., 65.

II. *Experiments and Results.*

In the experiments with anhydrous zinc sulphate there were determined the temperatures at which the reaction began and at which it became vigorous, and the residue was analyzed for sulphur. Heating was carried on in the tube-furnace. As it was necessary to carry off the volatile products of the reaction, a current of gas had to pass through the tube. Air being excluded on account of the rapid combustion of the carbon, nitrogen was used. It was prepared by Lupton's method,[15] in which ammonia gas diluted with air is conducted over a mixture of cupric oxide and metallic copper heated to redness in a small ignition-tube. The metallic copper is oxidized by the air and again reduced by the hydrogen of the ammonia. The gas-mixture is obtained by forcing air through a concentrated solution of ammonia. It will be seen below, from the slow burning of the carbon in the tests, that the nitrogen thus obtained was not pure. As this did not interfere with the work, no especial pains were taken to prepare pure nitrogen. Three runs were made with 6, 8 and 22 times as much carbon in the form of charcoal as was called for by the formula. The charge, about 0.4 g. of anhydrous zinc sulphate, was mixed and ground with the desired weight of charcoal and placed in a 3-in. porcelain-boat. The mode of operating was the same as in the previous tests made in the tube-furnace.

In the first test (6 times the theoretical quantity of carbon) the temperature of rapid reaction was first noted; it was 528° C. This is the temperature at which zinc sulphate begins to be decomposed when heated in air. The evolution of gas, however, was found to be much more energetic, showing that the decomposition by means of carbon at this temperature was more powerful than by means of heat alone. The temperature was then lowered; the evolution of sulphurous acid diminished correspondingly and ceased at 475° C. The residue was white, all the carbon had been oxidized. In run No. 2 (8 times the theoretical quantity of carbon) the lowest temperature at which any sulphur dioxide could be obtained was 409° C.; it ceased after 1 hour's heating. Upon raising the temperature to 425° C., sulphur dioxide was again liberated, but weakly, and lasted also

[15] *Chemical News*, 1876, xxxiii., 90.

only for 1 hour. Heating at 528° C. caused the reaction to show much activity; it lasted 13 hours; the residue was again found to be white. In order to check the results of the two preceding tests and to investigate the behavior between 425° and 528° C., a third run (22 times the theoretical quantity of carbon) was made. The first sulphurous gas was again detected at 409° C.; at 463° C. carbon acted upon the sulphate, but not strongly; the effect weakened quickly and was hardly perceptible after 2.5 hours; at 500° C. the reaction was fairly rapid, but weakened after a few hours; at 528° C. there was the usual marked increase in the evolution of gas; the reaction continued for 27 hours, weakening toward the end; all the carbon had been consumed. Upon a further addition of carbon and heating to 528° C. no further evolution of sulphur dioxide could be detected.

The weights taken in the runs were those of the anhydrous zinc sulphate, of the charcoal and of the residue. As the percentage of ash of the charcoal had been determined, the weight of the decomposed sulphate could be easily found. The residue was analyzed for sulphide and sulphate-sulphur by the method advocated by Blair.[16] The residue was dissolved in a 250-cc. decomposition-flask in a small amount of hydrochloric acid, and the hydrogen sulphide evolved conducted through an alkaline solution of lead nitrate held in a 140-cc. Erlenmeyer beaker. The hydrochloric acid solution held the sulphate-sulphur and the absorption-flask the lead sulphide precipitated by the hydrogen sulphide. The leading results of the three tests are brought together in Table XI.

TABLE XI.—*Decomposition of Zinc Sulphate by Heating with Carbon.*

Test Number	1.	2.	3.
Multiple of theoretical carbon	6	8	22
Time of heating—hours	7	13	27
Temperature—degrees Centigrade	475	528	528
Loss in weight of $ZnSO_4$—per cent	6.27	30.55	42.61
Sulphide S in residue of $ZnSO_4$[a]—per cent	trace	0.61	1.41
Sulphate S in residue of $ZnSO_4$[a]—per cent	17.80	6.85	2.06

[a] $ZnSO_4$ contains 19.86 per cent. of S.

[16] *The Chemical Analysis of Iron and Steel*, Philadelphia, 1901, p. 60.

III. *Summary.*

The reaction of carbon upon zinc sulphate begins at 409° C., but is weak and lasts only a short time; it grows stronger at 425°, 463°, 475°, 500° C., but weakens perceptibly after a few hours; only at 528° C. does carbon appear to become decidedly active, but even here the effect is at best very slow and imperfect, as it is opposed by the direct reduction of zinc sulphate to zinc sulphide. Under the most favorable conditions the 19.86 per cent. of sulphur of zinc sulphate are reduced to 3.47 per cent.; of this total, 1.41 per cent. are sulphide-sulphur and 2.06 per cent. sulphate-sulphur.

PART III. FORMATION OF ZINC SULPHATE BY ROASTING.

I. *Introduction.*

Blende in its deposits is usually associated with other metallic sulphides, such as galena, pyrite, marcasite, chalcopyrite, etc. If the minerals are sufficiently coarse, galena can be readily separated from blende by crushing and washing; pyrite and chalcopyrite, however, only very imperfectly. Magnetic concentration has, within a few years, come to the assistance of the common methods of ore-dressing, and does satisfactory work. (See "Roasting and Magnetic Concentration of a Blende-Marcasite Concentrate," later in this volume.) When metallic sulphides are very intimately mixed, ore-dressing methods fail and chemical methods come into play. The earliest of the latter class has been to give the ore a sulphatizing-roast,[17] aiming to oxidize iron and copper, and to sulphatize the lead and zinc and remove the zinc by leaching; but this has always been unsatisfactory. The only locality where it is carried out on a working-scale is at the Herzog-Julius and Frau Sophieen works in the Harz mountains.[18] Here, the leaching with water has to be supplemented by treatment with acid to dissolve basic zinc salts and zinc oxide. While no details are given as to the percentage of extraction, it may be inferred to be unsatisfactory from the fact that the slag of the blast-furnace in which the roasted and

[17] Ingalls, *Engineering and Mining Journal*, May 3, 1902.

[18] Bräuning, *Zeitschrift fuer Berg-, Huetten- und Salinen-Wesen in Preussen*, 1877, xxv., 144; Kerl, *Mineral Industry*, 1895, iv., 83; Schnabel, *Handbuch der Metallhuettenkunde*, 2d ed., Berlin, 1901, p. 312; Ingalls, *Metallurgy of Zinc and Cadmium*, New York, 1903, 677.

leached ore is smelted assays 19.12 per cent. of zinc. It becomes, therefore, of much value to study the conditions under which the maximum of sulphate is formed in roasting blende.

Plattner[19] is probably the earliest writer who investigated the behavior of blende when subjected to an oxidizing roast. He says that, if blende, practically free from iron and in a finely divided state, is roasted with stirring at a dark-red heat, the oxidation will proceed slowly, and the sulphur dioxide evolved will be converted in part into sulphur trioxide, this process being favored by a thick bed of ore. Some of the sulphur trioxide will combine with the zinc oxide formed to yield normal sulphate and tetrabasic sulphate, and some will have an oxidizing effect upon the blende still undecomposed. If the roast is carried on at an elevated temperature and with a strong draft, causing free access of air, less sulphur trioxide will be formed and hence less zinc sulphate. If the temperature reaches the point at which the tetrabasic sulphate is decomposed, the normal sulphate cannot well form, and the result is a product containing only zinc oxide. The formation of a tetrabasic zinc sulphate has already been discussed.

The presence of iron in blende as an isomorphous sulphide is known to have a retarding influence upon the roasting of blende. Minor[20] has published interesting remarks on this point, treating four samples of blende with from 2 to 15 per cent. of Fe. His results are given in part in Table XII.

TABLE XII.—*The Retarding Influence of Iron Upon the Roasting of Blende.*

Raw Blende.		Percentage of Sulphur in Roasting Blende.					
No. of Test.	Iron.	1 Hour.	2 Hours.	3 Hours.	4 Hours.	5 Hours.	6 Hours.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
1.....	2.33	10.32	2.86	1.80	1.71	0.95	0.08
2.....	13.12	24.33	21.95	20.65	17.02	2.15	0.18
3.....	2.06	14.69	6.27	1.23	0.22	0.16	0.06
4.....	15.94	27.25	24.30	18.25	11.94	4.27	0.04

The roasts are seen to have been carried very far. The behavior of the samples low in iron, Nos. 1 and 3, is normal, the

[19] *The Metallurgischen Roestprocesse*, Freiberg, 1856, pp. 87, 142.

[20] *Chemiker Zeitung*, 1889, 1602; *Berg- und Huettenmaennische Zeitung*, 1889, 446.

percentage of sulphur falling off quickly in the first two hours, and then more slowly. With the blende high in iron, samples Nos. 2 and 4, the percentage of sulphur decreases extremely slowly, and then drops suddenly in the fifth hour; at the end of the sixth hour the blende with 15.98 per cent. (or nearly 16) of Fe contains less sulphur than any of the other three samples. An explanation for this peculiar behavior is not given.

With regard to ferruginous blende, Jensch[21] makes the surprising statement that when roasted sweet, the sulphur still present is almost wholly combined with iron as monosulphide. This is contrary to the general belief based upon practical experience, and the fact that iron sulphide is more readily oxidized than zinc sulphide; contrary to the researches of Plattner,[22] who found that, under normal conditions, the iron was oxidized to ferric oxide, and that, when the temperature was very high and the access of air limited, magnetic oxide was readily formed; and contrary to the researches of Prost,[23] who finds that part of the iron oxide formed combines with zinc oxide (see Appendix, Part IV.). The results of Jensch are, however, given in full in Table XIII to show that it is possible to roast dead an iron-bearing blende and leave part of the iron in the form of iron sulphide. Unfortunately, the details of operating are not given.

The progressive desulphurization in large-scale work is discussed in detail by Ingalls, to whose book[24] the reader may be referred.

II. *Description of Blende.*

The experiments on the formation of zinc sulphate were made with blende from Joplin, Mo., and from Warren, N. H.; some work was done with a sample from New Mexico. The analyses made are given in Table XIV.

The Joplin blende was massive; it had the characteristic brown color and resinous luster. It was chosen on account of its freedom from gangue and iron. The mineral from Warren,

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[21] *Zeitschrift fuer Angewandte Chemie*, 1894, 50; *Berg- und Huettenmaennische Zeitung*, 1894, 299.

[22] *Op. cit.*, p. 143.

[23] *Bulletin de l'Association Belge des Chimistes*, x., 246; through Ingalls, *Metallurgy of Zinc and Cadmium*, New York, 1903, 32.

[24] *Metallargy of Zinc and Cadmium*, New York, 1903, pp. 45 *et seq.*

TABLE XIII.—*Jensch's Results of Roasting Ferruginous Blende.*

		Upper Silesia.			Sweden.			Erzgebirge.	Styria.	
		a.	b.	c.	a.	b.	c.	a.	a.	b.
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
I. Raw blende.	Zn.	32.64	22.58	32.07	31.66	35.92	32.45	44.60	44.95	42.80
	Fe.	5.95	7.12	4.66	10.53	8.64	10.75	4.29	0.72	0.97
	Pb.	3.12	3.20	2.88	6.88	11.10	5.38	2.50	1.12	0.85
	Mn.	.....	.....	.....	.....	1.30	0.10	.....	.....	.....
II. Roasted blende.	Zn.	48.60	30.04	39.81	35.22	40.72	34.90	51.70	50.10	49.66
	Fe.	8.11	10.00	5.73	11.78	9.17	11.62	5.12	0.84	1.16
	Pb.	3.95	4.19	3.31	7.24	12.04	5.68	21.86	1.20	0.97
	Mn.	.....	.....	.....	.....	1.46	.....	.....	.....	.....
III. Leached* residue from II.	Zn.	0.42	0.30	0.58	0.26	0.46	0.18	0.22	.....	0.18
	S.	1.64	1.83	2.02	1.30	2.40	1.06	1.29	1.36	0.68
	ZnS.	0.62	0.45	0.87	0.39	1.69	0.27	1.33	.....	0.27
	S available for iron.	1.44	1.68	1.73	1.17	2.17	0.98	1.18	1.36	0.59
	FeS.	3.96	4.62	4.74	3.22	5.97	2.69	3.24	0.99	1.62
IV. Sulphur calculated as in combination with	Zn.	0.20	0.15	0.29	0.13	0.23	0.08	0.11	.....	0.09
	Fe.	1.44	1.68	1.73	1.17	2.17	0.98	1.18	1.36	0.59
V. Proportion of	Zn : Fe.	1 : 7	1 : 11	1 : 6	1 : 9	1 : 10	1 : 12	1 : 11	1 : ∞	1 : 6.5

* Roasted blende (II.) was leached with water and ammonium acetate, to dissolve zinc and lead sulphate, and boiled with caustic soda, to dissolve zinc oxide and silicate, leaving as residue: zinc, iron and sulphur.

TABLE XIV.—*Analyses of Blende.*

	Joplin, Mo.	Warren, N. H.	New Mexico.
	Per Cent.	Per Cent.	Per Cent.
Zn.....	66.32	53.94	.....
Fe.....	0.45	8.80	13.40
S.....	32.54	31.37	.....
Pb.....	.....	3.82	.. ..
SiO_2.....	not determined	0.82	.....
Cd.....	not determined	trace	.....

N. H., was the best representative of blende high in iron that could be obtained in considerable quantity. The sample was picked over at the mine to obtain the ore as pure as possible. When crushed through an 8-mesh sieve in the laboratory, it was again hand-sorted. Nevertheless, some lead remained behind, as shown by the analysis. The ore was made up of crystalline fragments; it had a brownish-black color and a luster more adamantine than resinous. The blende from New Mexico, used in a few cases to see how it differed in its behavior from Warren blende, was a concentrate. The mineral was black and had an adamantine luster. It was carefully picked over to exclude all foreign matter.

III. *Outline of Operations.*

The temperatures at which different specimens of blende decrepitate and ignite are of value in an investigation of the mineral; hence these were determined. The ore to be given a sulphatizing-roast was passed through 8- and 40-mesh sieves; a charge weighed from 200 to 350 g.; the roastings were carried on in the gas muffle-furnace; temperature and time were varied with the object of obtaining the maximum normal sulphate; the temperatures were measured as in the experiments for formation of basic salts (p. 826); and samples for analysis were taken at intervals of from 0.5 to 1.5 hours. Four determinations were made from each sample: Total zinc, by titration with potassium ferrocyanide; total sulphur, by fusion with potassium nitrate and precipitation with barium chloride; normal zinc sulphate, by leaching with warm water (8 decantations of 20 cc. being sufficient) and titration of zinc as in the determination of total zinc; and sulphur in basic zinc sulphate, the method of Steinbeck[25] being employed (covering of sample with sodium bicarbonate, adding enough water for solution, allowing to stand for 24 hours with occasional stirring, filtering, acidifying and boiling filtrate, and precipitating with barium chloride). Beside the regular sulphatizing-roasts, special tests were made with Joplin blende by mixing it raw and dead-roasted with pyrite in order to increase the percentage of soluble sulphate. In this series all the samples taken were analyzed for normal zinc sulphate, and only those showing a maximum of soluble sulphate were analyzed for total zinc, total sulphur and sulphur as basic sulphate.

Attention has been called by Prost[26] to the probable existence of a zinc ferrate, $ZnFe_2O_4$, formed in roasting ferruginous blende, which is insoluble in dilute sulphuric acid, and may be the cause of the unsatisfactory results obtained by sulphatizing-roasts. Several experiments were made to determine whether this compound existed, and, if this was found to be true, to find the conditions that favored its formation.

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[25] "Chemisch-Analytische Untersuchungen über die Veraenderungen, welche der Mansfelder Kupferstein bei der Roestung erleidet." *Zeitschrift fuer Berg-, Huetten- und Salinen Wesen in Preussen*, 1863, xi., 123.

[26] *Op. cit.*

IV. *Details of Experiments.*

1. *Decrepitation.*—A small quantity of medium-size pieces was picked from each of the three lots of blende and crushed to pass an 8-mesh sieve. Each sample was then placed in a large boat and heated in the electric retort-furnace, with the junction of the thermo-couple in contact with the ore. The neck of the retort was kept open to permit watching the interior. The Warren blende did not decrepitate at all. The behavior of the blende from Joplin and from New Mexico was very similar, so they can be discussed together. Decrepitation began at 290° C. and lasted a few minutes. The temperature was then raised 10° at a time. With every rise decrepitation started again and did not cease when the ignition-point had been reached. When it had stopped, about 25 per cent. of the charge was very fine and the rest medium coarse, but many pieces had retained about their original size, i.e., they were caught on a 12-mesh sieve.

2. *Ignition-Point.*—Three or more tests were made in the electric tube-furnace with each sample, to fix the ignition-point. The sample was placed in a boat, the mode of operating was the same as with the beginning of decomposition of zinc sulphate. In the preliminary tests, in which the furnace was heated rapidly to locate approximately the point, the ignition-temperature was found to be about 40° C. too high. In the regular tests the furnace was brought slowly to near the ignition-point and the temperature then raised at the rate of a degree to a degree and a half per minute. The tests on separate samples of decrepitated Joplin blende gave the ignition-points, 480°, 479° and 482°, or an average of 480° C. Of the Warren blende, which did not decrepitate, 3 tests were made on coarse (through 8- and on 12-mesh sieve) and 3 on fine (through 80-mesh sieve) material. The tests on the former gave, 557°, 557° and 556°, or an average of 557° C. as the ignition-point; the latter, 518°, 513° and 515°, or 515° C. as an average. The difference of 42° C. is, of course, due to the fine material being more easily heated through than the coarse. The blende from New Mexico gave, 535°, 531° and 536°, or an average of 534° C.

Table XV shows that the ignition of ferruginous blende lies higher than when running low in iron, and that, as far as the three samples are concerned, the rise is nearly proportional to

TABLE XV.—*Ignition-Point of Blende.*

Source of Blende.	Joplin, Mo.	Warren, N. H.	New Mexico.
Size of blende	coarse and fine	fine	coarse and fine
Percentage of iron	0.45	8.80	13.40
Ignition-point, degrees centigrade	480	515	534

the iron, it being 4.19° C. for an increase of 1 per cent. of iron. If the samples are to be roasted, they have to be brought to above their ignition-temperatures, as none of them appeared to have been decomposed. The light Joplin blende would have shown any outward change more readily than the two dark samples, but it looked just as bright and shiny as it did before it was heated. Something similar has been more recently noticed by Eakle and Sharwood.[27]

3. *Sulphatizing-Roast of Joplin Blende.*—With the ignition-point of Joplin blende at 480° C., it was thought that a satisfactory start in sulphatizing could be made at a temperature about 200° higher. Three roasts were carried out in the gas muffle-furnace at 690°, 740° and 790° C., or 50° C. below, at and 50° C. above, the lowest temperature of complete decompositlon of zinc sulphate. The results are given in Tables XVI (690° C.), XVII (740° C.) and XVIII (790° C.).

With regard to Table XVI, the absence of data for basic salt during the first 6.5 hours is due to an error in manipulation, in that the samples were leached with sodium acetate after they had been freed from normal sulphate with warm water; from the remaining three samples the basic salt was extracted by treating with sodium bicarbonate. As the samples of ore were used up and the data would have no practical value, the experiment was not repeated.

The elimination of sulphur at this low temperature is regular, but very slow, the amount having fallen in 12.5 hours from 32.54 to 14.08 per cent. The percentage of normal sulphate increases gradually and reaches its maximum with 0.623 per cent., while the basic sulphate is still increasing, at the end of the test, after 12.5 hours.

[27] *Engineering and Mining Journal*, June 23, 1904.

TABLE XVI.—*Results of Roasting Joplin Blende.*

[Analysis: Zn, 66.32; S, 32.54; Fe, 0.45 per cent. Charge: 350 g.; size, 8-mesh; temperature, 690° C.; time, 12.5 hours.]

	Actual Analyses.								
Sample taken after.....	Hrs. 1	Hrs. 2	Hrs. 3	Hrs. 4	Hrs. 5	Hrs. 6.5	Hrs. 8.5	Hrs. 10.5	Hrs. 12.5
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Total zinc.....	66.85	67.67	68.70	69.98	70.82	71.02	71.03	71.55	72.06
Total sulphur.....	30.35	28.47	25.68	23.71	21.52	19.42	19.03	17.48	15.30
Zinc as normal sulphate.....	0.155	0.199	0.206	0.290	0.335	0.409	0.463	0.672	0.638
Sulphur as basic sulphate....	*a*	*a*	*a*	*a*	*a*	*a*	0.103 *b*	0.121 *b*	0.143 *b*
Analyses Corrected to the Basis of 66.32 Per Cent. of Zinc.									
Total sulphur.....	30.09	27.90	24.78	22.47	20.15	18.18	17.76	16.20	14.08
Zinc as normal sulphate.....	0.154	0.195	0.199	0.274	0.313	0.382	0.432	0.623	0.587
Sulphur as normal sulphate	0.077	0.098	0.100	0.137	0.156	0.191	0.216	0.312	0.294
Sulphur as basic sulphate....	*a*	*a*	*a*	*a*	*a*	*a*	0.096 *b*	0.112 *b*	0.134 *b*
Sulphur as sulphide-sulphur	30.01	27.80	24.63	22.33	20.00	17.94	17.29	15.78	13.65

a Ore leached with sodium acetate which did not dissolve any basic sulphate; no sample left to repeat test with sodium bicarbonate.

b Ore leached with sodium bicarbonate, which decomposed basic sulphate.

c Calculated.

TABLE XVII.—*Results of Roasting Joplin Blende.*

[Analysis: Zn, 66.32; S, 32.54, Fe, 0.45 per cent. Charge: 350 g.; size, 8-mesh; temperature, 740° C.; time, 15 hours.]

	Actual Analyses.									
Sample taken after.....	Hrs. 1.5	Hrs. 3	Hrs. 4.5	Hrs. 6	Hrs. 7.5	Hrs. 9	Hrs. 10.5	Hrs. 12	Hrs. 13.5	Hrs. 15
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Total zinc.....	70.54	73.08	74.43	74.98	74.87	74.92	74.44	74.49	74.61	76.17
Total sulphur.....	24.79	17.22	12.66	7.75	6.28	6.16	5.70	4.72	3.96	3.47
Zinc as normal sulphate.....	0.108	0.315	0.733	1.23	1.56	1.27	2.13	2.42	2.78	2.35
Sulphur as basic sulphate...	0.011	0.046	0.159	0.252	0.583	0.777	0.823	0.857	0.867	0.727
Analyses Corrected to the Basis of 66.32 Per Cent. of Zinc.										
Total sulphur.....	23.30	15.63	11.42	6.85	5.50	5.45	5.08	4.20	3.53	3.02
Zinc as normal sulphate.....	0.101	0.285	0.662	1.09	1.38	1.10	1.89	2.16	2.48	2.04
Sulphur (*a*) as normal sulphate.....	0.051	0.148	0.331	0.545	0.690	0.550	0.945	1.08	1.24	1.02
Sulphur as basic sulphate...	0.010	0.042	0.144	0.223	0.517	0.687	0.733	0.763	0.770	0.633
Sulphur (*a*) as sulphide-sulphur.....	23.24	15.44	10.94	6.08	4.29	4.21	3.40	2.36	1.52	1.37

a Calculated.

Table XVII shows that, with the higher temperature of 740° C., the decrease in total sulphur is more rapid than with 690° C., sinking to 4.2 per cent. after 12 hours, and to 3.02 per cent. after 15 hours; the fall is quicker at the beginning than at the end; the low percentage of sulphide-sulphur at the end of the roast shows that the blende has been well decomposed. On account of the larger amount of sulphur oxidized, the percentage of normal sulphate found is also greater than before; with the exception of the time-interval between the seventh and tenth hour the increase is regular and reaches a maximum after 13.5 hours. The basic salt forms less rapidly than the normal salt and attains its maximum at the same time as the normal salt.

TABLE XVIII.—*Results of Roasting Joplin Blende.*

[Analysis: Zn, 66.32]; S, 32.54; Fe, 0.45 per cent. Charge: 350 g.; size, 8-mesh; temperature, 790° C.; time, 7 hours.]

	Actual Analyses.				
Sample taken after.....	1 Hour.	2.5 Hours.	4 Hours.	5.5 Hours.	7 Hours.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Total zinc.....	66.90	75.82	76.07	76.10	77.20
Total sulphur.....	27.27	10.49	3.10	2.16	1.16
Zinc as normal sulphate.....	0.049	0.170	.....	1.38	0.93
Sulphur as basic sulphate.....	0.012	0.033	.....	0.760	0.480
Analyses Corrected to the Basis of 66.32 Per Cent. of Zinc.					
Total sulphur.....	25.28	9.17	2.70	1.88	0.997
Zinc as normal sulphate.....	0.047	0.152	.....	1.20	0.799
Sulphur ([a]) as normal sulphate.....	0.024	0.076	.....	0.60	0.40
Sulphur as basic sulphate.....	0 011	0.029	.....	0.66	0.41
Sulphur ([a]) as sulphide-sulphur.....	25.24	9.06	.....	0.62	0.19

[a] Calculated.

Table XVIII shows that, with a temperature of 790° C., the roast could be stopped after 7 hours, the total sulphur having fallen to 0.997 per cent. and the sulphide-sulphur to 0.19 per cent. As the roast was carried on above the temperature at which the normal sulphate is completely decomposed, the maximum soluble sulphate is reached early and the total is smaller than in the preceding test. That the basic sulphate should follow a similar path was to be expected.

A difficulty encountered in leaching some of the samples

which were afterward discarded may be mentioned. They were treated for a long time with hot water, the waters remaining in contact with the ore for several hours before they were decanted; the dissolved zinc was determined in the decantations obtained in 24 hours. After leaching 5 days in this manner without removing all the soluble zinc, the ore was boiled for 36 hours with occasional decantations and still some zinc salt went into solution. The explanation is found in the fact that basic sulphates are slightly soluble in hot and boiling water.[28]

Comparing the three roasts as plotted in Fig. 5, it is seen that the elimination of total sulphur increases with the temperature, while the time required decreases. The curves for sulphide-sulphur (or undecomposed blende) keep close to the curves for total sulphur, remaining, of course, below them. The narrow space between a pair of lines shows how small is the quantity of sulphate formed. The largest amount of normal zinc sulphate readily soluble in water is obtained at 740° C.; leaving out the jog in the curve at the ninth hour, the progress is quite regular until the maximum is reached, but even at best the total of normal zinc sulphate is extremely small. Basic salts, practically insoluble in water, keep an even pace with the normal salt, more so with high (790° C.) and low (690° C.) temperatures than under the favorable conditions of 740° C.

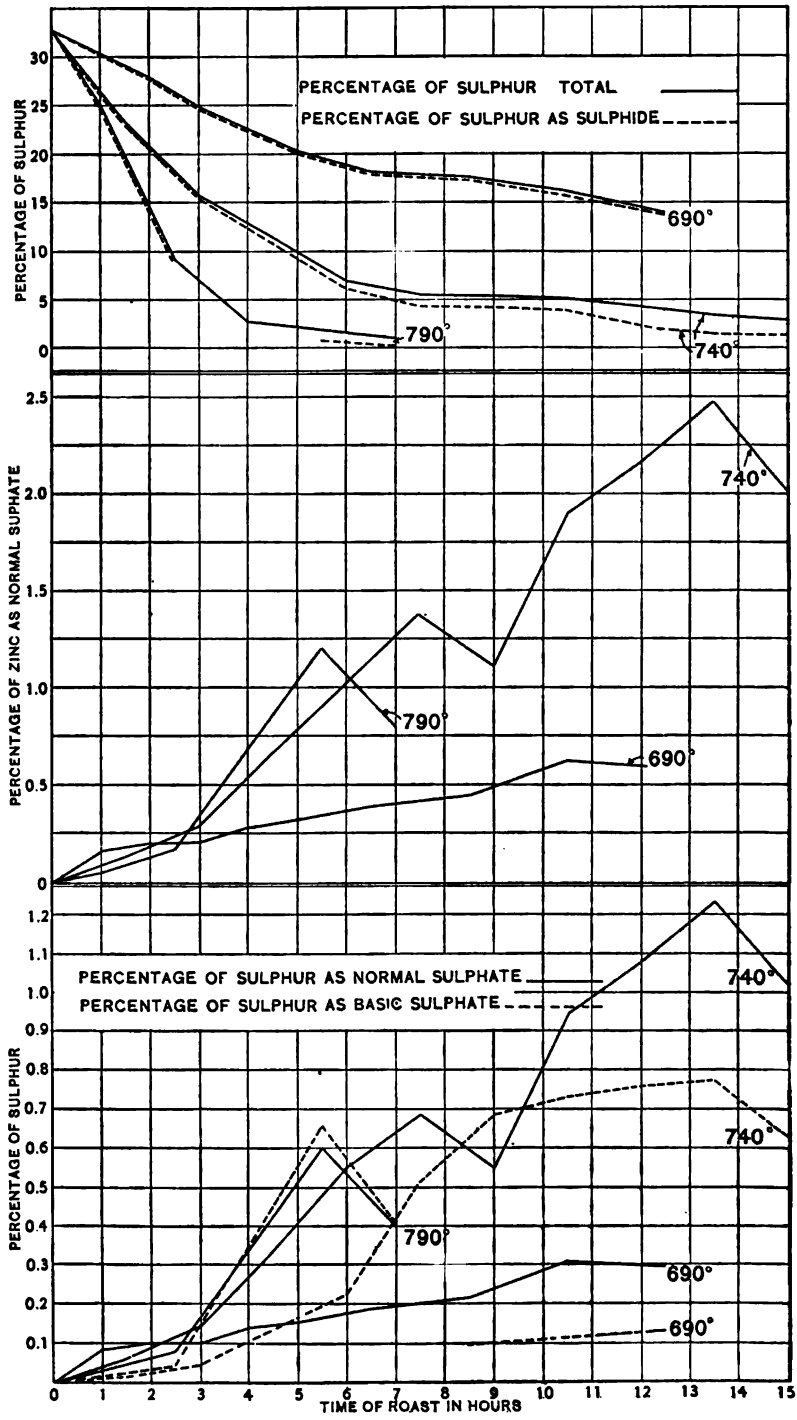
4. *Sulphatizing-Roast of Warren Blende.*—The blende from Warren, N. H., contains 3.82 per cent. of lead. Part of the galena is so finely disseminated through the blende that it is impossible to remove it by picking, even when using a magnifying glass. The presence of 4.41 per cent. of galena complicates matters. It is known that in roasting galena slowly at a low temperature more sulphate is formed than in roasting quickly at a high temperature;[29] hence, as far as the galena is concerned, the latter is preferable. As 690° C., with easy-roasting blende, had given a very small percentage of soluble zinc sulphate, 790° and 900° C. were chosen as roasting-temperatures. The amount of lead sulphate formed is liable to be increased[30] by

[28] Vogel, *Journal fuer Chemie und Physik*, edited by J. S. C. Schweigger, 1814, xi., 416.

[29] Hofman, *Metallurgy of Lead*, New York, 1899, p. 12.

[30] Plattner, *Die Metallurgischen Roestprocesse*, Freiberg, 1856, p. 45.

FIG. 5.



THE ROASTING OF BLENDE FROM JOPLIN, MO.

the sulphur trioxide set free in roasting the blende. Assuming a quick roast to take place according to the formula,

$$3\ PbS + 10\ O = 2\ PbO + PbSO_4 + 2\ SO_2,$$

it is fair to suppose that the proportion of $PbO : PbSO_4$ will be as 1 : 1 at 790° C. and slightly less at 900° C. The results obtained in the two roasts are given in Tables XIX (790° C.) and XX (900° C.), and graphically represented in Fig. 6.

A study of the data in Table XIX shows that the elimination of total sulphur is regular, i.e., quick at the beginning and slow later on, but unsatisfactory, as after 15 hours it has been reduced to only 5.61 per cent. The percentage of normal zinc sulphate formed rises very quickly, reaches a maximum after 7.5 hours and then falls very gradually. The percentages of normal and basic sulphate-sulphur apparently keep an even pace, showing that under favorable conditions for normal sulphate basic sulphate forms in considerable amounts. This explains the necessity for acid-treatment in the Harz mountains after leaching with water.

TABLE XIX.—*Results of Roasting Warren Blende.*

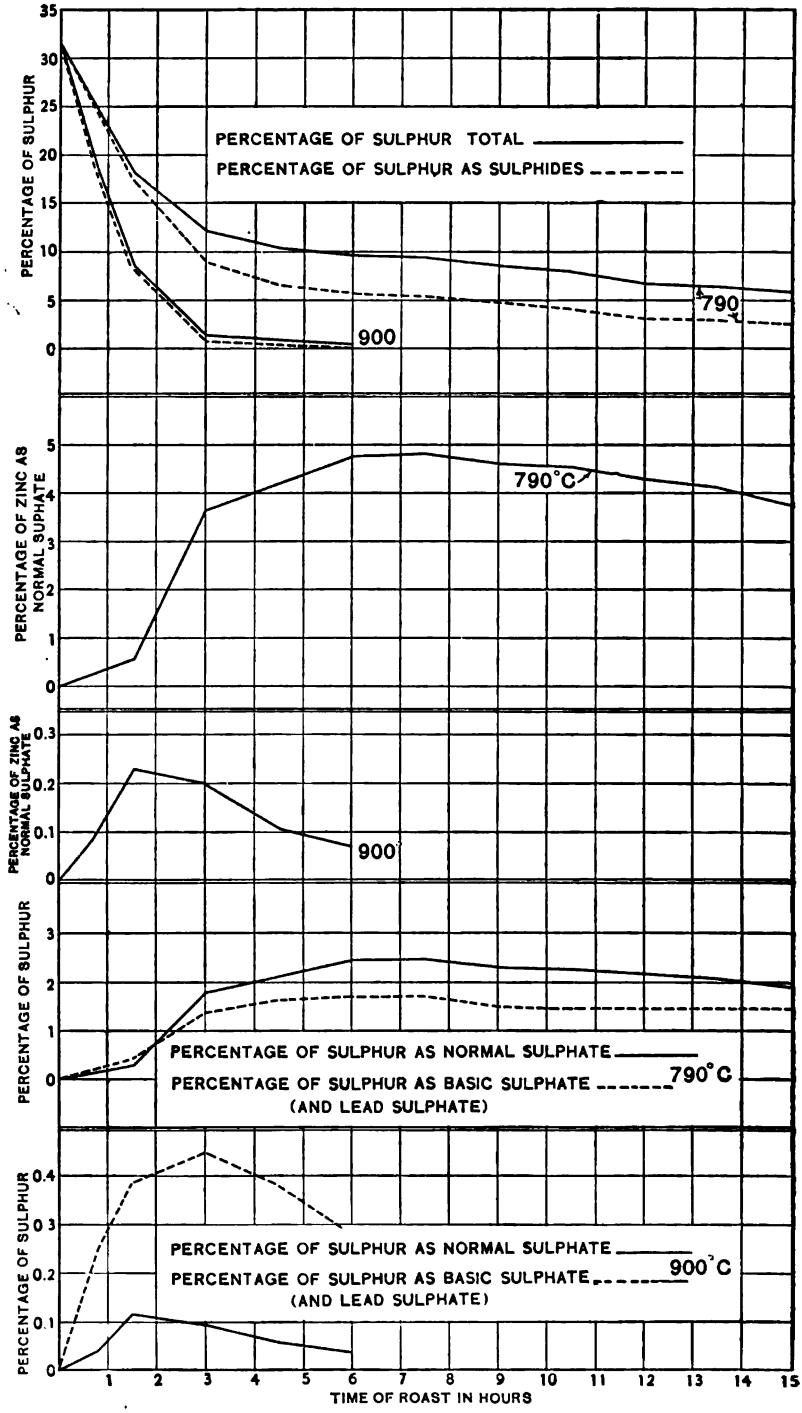
[Analysis: Zn, 53.94; S, 31.37; Fe, 8.80; Pb, 3.82; SiO_2, 0.81 per cent.; Cd, trace. Charge: 300 g.; size, 8-mesh; temperature, 790° C.; time, 15 hours.]

	Actual Analyses.									
Sample taken after..	Hrs. 1.5	Hrs. 3	Hrs. 4.5	Hrs. 6	Hrs. 7.5	Hrs. 9	Hrs. 10.5	Hrs. 12	Hrs. 13.5	Hrs. 15
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Total zinc.....	58.56	56.15	56.36	56.28	55.97	55.82	55.71	55.09	55.37	55.24
Total sulphur.....	19.56	12.58	10.79	10.07	9.75	8.66	7.98	6.65	6.42	5.96
Zinc as normal sulphate.....	0.597	3.76	4.41	5.06	5.08	4.74	4.68	4.37	4.21	3.95
Sulphur as basic sulphate...	0.400	1.41	1.66	1.75	1.76	1.55	1.49	1.51	1.49	1.49
Analyses Corrected to the Basis of 53.94 Per Cent. of Zinc.										
Total sulphur.....	18.02	12.09	10.31	9.65	9.40	8.37	7.68	6.51	6.25	5.61
Zinc as normal sulphate.....	0.550	3.61	4.22	4.85	4.89	4.58	4.53	4.27	4.10	3.72
Sulphur (a) as normal sulphate.....	0.275	1.81	2.11	2.43	2.45	2.29	2.27	2.14	2.05	1.86
Sulphur as basic sulphate...	0.367	1.35	1.59	1.68	1.69	1.49	1.45	1.48	1.46	1.40
Sulphur (a) as sulphide-sulphur.....	17.34	8.93	6.61	5.54	5.26	4.59	3.96	2.89	2.74	2.35

[a] Calculated.

The results in Table XX show that at 900° C. the percentage of total sulphur falls in 3 hours from 31.37 to 1.16 per cent.,

FIG. 6.



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and is reduced in 6 hours to 0.48 per cent. (= 0.16 per cent. sulphide-sulphur). The amount of normal zinc sulphate formed at this high temperature must be low; the insignificant figure of 0.228 per cent. forms the maximum, and is reached after 1.5 hours. The evil effect of a high temperature is seen by the sulphur of the basic sulphate being always higher than that of the normal sulphate, increasing up to the third hour and then diminishing.

TABLE XX.—*Results of Roasting Warren Blende.*

[Analysis: Zn, 53.94; S, 31.37; Fe, 8.80; Pb, 3.82; SiO_2, 0.81 per cent.; Cd, trace. Charge: 300 g.; size, 8-mesh; temperature, 900° C.; time, 6 hours.]

	Actual Analyses.				
Sample taken after.....	0.75 Hrs.	1.5 Hours.	3 Hours.	4.5 Hours.	6 Hours.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Total zinc.....	58.06	60.43	63.66	63.96	64.48
Total sulphur.....	19.74	9.62	1.37	0.98	0.58
Zinc as normal sulphate.....	0.085	0.255	0.232	0.125	0.082
Sulphur as basic sulphate.....	0.266	0.433	0.527	0.450	0.340
Analyses Corrected to the Basis of 53.94 Per Cent. of Zinc.					
Total sulphur.....	18.35	8.59	1.16	0.83	0.48
Zinc as normal sulphate.....	0.079	0.228	0.196	0.105	0.068
Sulphur (a) as normal sulphate.....	0.040	0.114	0.098	0.053	0.034
Sulphur as basic sulphate.....	0.248	0.387	0.448	0.380	0.285
Sulphur (a) as sulphide-sulphur....	18.06	8.09	0.61	0.40	0.16

[a] Calculated.

A comparison of the two roasts in Fig. 6 shows for 790° C. clearly the imperfect elimination of total sulphur in 15 hours' roasting, coupled with a better formation of soluble zinc sulphate than is the case with 900° C. Lengthening the time of roasting to remove more sulphide-sulphur is unfortunately accompanied by a gradual but sure decomposition of the normal salt. This explains, perhaps, why it is that in the Harz mountains it has been found necessary to roast an ore repeatedly at a low temperature to obtain any commercial extraction whatever; a single prolonged roast would give unsatisfactory results. At 790° C. the basic sulphate curve is practically always beneath that of the normal sulphate; with 900° C. just the reverse is the case.

Comparing the behavior of the Warren blende (Fig. 6) with that of the Joplin blende (Fig. 5), it is seen that in regard to the elimination of total sulphur, the behavior of Warren blende at 790° C. corresponds quite closely to that of Joplin blende at 740° C.; a resemblance is also to be found between Warren blende at 900° C. and Joplin blende at 790° C. In other words, the ferruginous blende gives up its sulphur less easily than the blende that runs low in iron. It may be that the geological age of the formation, as suggested by Jensch,[31] also has something to do with it, as Warren blende occurs in quartz-veins in Archaean gneiss, while Joplin blende is found in Carboniferous limestone. The irony blende under suitable conditions furnishes more soluble zinc sulphate than blende low in iron, and the formation extends over a longer roasting-period. This is to be expected, considering that iron sulphide is converted into normal ferrous sulphate which at 590° C. begins to be changed into ferric oxide, passing through stages of basic sulphate decomposed only at a higher temperature, all of which give off sulphur trioxide, which has an oxidizing effect upon zinc sulphide and a sulphatizing effect upon zinc oxide. The sulphatizing effect of basic iron sulphate is brought out also by the quick rate of decomposition of the soluble zinc sulphate with the Joplin blende when the maximum has been reached in comparison with the blende from Warren. In agreement with the larger amount of normal zinc sulphate formed with blende rich in iron, stands the fact that it forms more basic zinc sulphate than blende low in iron.

5. *Sulphatizing-Roast with Addition of Pyrite.*—Starting with the fact proved by the preceding tests that a ferruginous blende gave more soluble zinc sulphate than blende low in iron, it was thought that an addition of pyrite to Joplin blende might greatly improve matters and furnish a satisfactory percentage of normal sulphate. Thus, a concentrate of galena, blende and pyrite, when subjected to a sulphatizing-roast, might furnish after leaching with water a desirable smelting ore of lead. Two tests were carried out, one with raw blende, the other with dead-roasted blende, to study the sulphatizing effect of pyrite under two opposite conditions. It was hoped that by furnishing to the sulphur trioxide a large quantity of zinc oxide on which it

[31] *Loc. cit.*

could act, a considerable quantity of soluble-sulphate would be formed, as sulphur trioxide acts more strongly upon zinc oxide than upon zinc sulphide. The pyrite used came from the Davis mine in western Massachusetts. It was very pure, the analysis giving 52.95 per cent. of sulphur or only 0.44 per cent. less than that required by the formula. In considering the quantity of pyrite to be added to the blende, the plan was to have twice the amount required by theory, i.e., 2 parts of sulphur to 1 part of zinc, or 120 g. pyrite for 100 g. blende and for 83.8 g. zinc oxide. It was expected that one atom of sulphur of the pyrite starting to burn below the ignition-point of the blende would have practically no sulphatizing effect, but that the remaining monosulphide would accomplish the desired result. The temperatures chosen were 790° and 650° C., i.e., above and below 740° C., the temperature at which Joplin blende had given the maximum of soluble salt. Heating at 790° C., when zinc sulphate is rapidly decomposed, would show how much the sulphatizing effect of pyrite could counteract this tendency; the temperature of 650° C., being high enough to decompose ferrous sulphate rapidly but zinc sulphate only very slowly, would show whether, with very incomplete desulphurization of blende, the oxidized part would be sulphatized by the pyrite. The results are recorded in Tables XXI, XXII and XXIII.

The data in Table XXI were obtained by roasting a mixture of blende and pyrite ground to 8-mesh size, the temperature being raised very slowly in order to avoid mechanical loss by decrepitation and to permit control of temperature when the pyrite began to burn. This occurred at about 500° C. The temperature rose in half an hour to 790° C., when the gas was shut off; it remained constant for several minutes and then began to fall, necessitating the turning on of the gas. With the ignition of the pyrite, dense white fumes arose and continued for half an hour, when the pyrite ceased to glow. The first sample was taken half an hour after the ore had reached 790° C.; the others as shown in the headings. The results in Table XXI show a steady increase of soluble zinc sulphate, but, after 6.5 hours, the total had reached only 2.15 per cent. corrected weight; it could not have risen much higher, as the total sulphur had been reduced to 3.66 per cent.; hence the roast was not carried any farther. The total amount of sul-

phate, however, is twice as large as when the raw blende was roasted alone at 790° C., and that of basic sulphate very much lower; both facts show that the addition of pyrite has a beneficent effect.

TABLE XXI.—*Results of Roasting Joplin Blende and Pyrite.*

[Charge: 100 g. Joplin blende and 120 g. Davis pyrite; temperature, 790° C.; time, 6.5 hours. Calculated analysis: Zn, 30.14; S, 43.68; Fe, 25.43 per cent.]

Size of ore, 8-mesh.

Sample taken after	0.5 Hours.	2 Hours.	3.5 Hours.	5 Hours.	6.5 Hours.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Zinc as normal sulphate	0.42	1.00	2.19	2.33	2.70

Analysis of Ore at Maximum Normal Sulphate.

	Zinc.	Sulphur.	Zinc as Normal Sulphate.	Sulphur as Normal Sulphate.	Sulphur as Basic Sulphate.	Sulphur as Sulphide-Sulphur.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Actual	37.83	4.59	2.70	1.35	0.831	2 41
Corrected to 30.14 per cent. of zinc	30.14	3.66	2.15	1.08	0.663	1.92

Size of Ore, 40-Mesh.

Sample taken after	0.5 Hours.	2 Hours.	3.5 Hours.	5 Hours.	6.5 Hours.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Zinc as normal sulphate	1.32	3.27	3.18	2.75	1.79

Analysis of Ore at Maximum Normal Sulphate.

	Zinc.	Sulphur.	Zinc as Normal Sulphate.	Sulphur as Normal Sulphate.	Sulphur as Basic Sulphate.	Sulphur as Sulphate-Sulphur.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Actual	39.92	2.85	3.27	1.64	0.868	0.34
Corrected to 30.14 per cent. of zinc	30.14	2.15	2.47	1.24	0.657	0.25

The second set of results in Table XXI was obtained by roasting a mixture of blende and pyrite ground to pass a 40-mesh screen, the other conditions being exactly the same as in

the first set of experiments. This fine grinding was made in order to have a finer material and a more intimate contact between particles than was possible with the previous charge of 8-mesh grains. The effect is striking in that the maximum normal sulphate, only slightly higher than with the coarser material (2.47 per cent. *vs.* 2.15 per cent.), is reached after 2 hours, and that the total sulphur at the same time reduced to a lower figure (3.66 per cent. *vs.* 2.15 per cent.). The ratios of sulphur as basic sulphate to sulphur as normal sulphate of the 8-mesh and 40-mesh samples at maximum normal sulphate do not differ much, viz., 0.663 : 1.08 = 0.61 for the coarse, and 0.657 : 1.24 = 0.53 for the fine; they are, however, about one-half of that obtained when the 8-mesh blende is roasted by itself, and show that in the latter case much more basic sulphate is formed than in the former.

The data in Table XXII were obtained from a charge of 100 g. Joplin blende roasted dead in 2 hours at 950° C., the sulphur being reduced to 0.085 per cent. The roasted ore was cooled, the theoretical quantity (83.8 g.) weighed out and mixed with pyrite of 8-mesh size. The percentage of normal sulphate increased with the time of roasting. It reached, however, only 2.50 per cent. after 3 hours, when further heating was stopped. It could not have gone much higher by prolonging the heating, as the total sulphur had already been reduced to 2.13 per cent.

The conditions in this roast, given in the lower part of Table XXII, were the same as those in the upper part, excepting that the dead-roasted blende and pyrite had been crushed together to pass a 40-mesh sieve. The effect of fineness of ore and consequent intimate contact of particles is as apparent here as it was with raw blende and pyrite in Table XXI. The maximum of soluble zinc sulphate, 5.18 per cent., is reached after 2 hours with the total sulphur reduced to 3.57 per cent. and the sulphide-sulphur to 0.07 per cent. It is a great deal higher than with the 8-mesh mixture of Table XXII; with that, some iron sulphide must have remained undecomposed, as the corrected analysis shows 0.57 per cent. of sulphur. The ratios of sulphur as basic sulphate to sulphur as normal sulphate (0.25 for 8-mesh, and 0.35 for 40-mesh, mixture) are lower than those of 790° C., or a smaller amount of basic sulphate is formed at

TABLE XXII.—*Results of Roasting Dead-Roasted Blende and Pyrite.*

[Charge: 83.8 g. dead-roasted Joplin blende and 120 g. Davis pyrite; temperature, 650° C.; time, 3 hours. Calculated analysis: Zn, 32.54; S, 31.18; Fe, 27.43 per cent.]

Size of ore, 8-mesh.

Samples taken after	1 Hour.	2 Hours.	3 Hours.
Zinc as normal sulphate	Per Cent. 1.50	Per Cent. 2.48	Per Cent. 3.12

Analysis of Ore at Maximum Normal Sulphate.

	Zinc.	Sulphur.	Zinc as Normal Sulphate.	Sulphur as Normal Sulphate.	Sulphur as Basic Sulphate.	Sulphur as Sulphide-Sulphur.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Actual	40.70	2.67	3.12	1.56	0.387	0.72
Corrected to 32.54 per cent. of zinc	32.54	2.13	2.50	1.25	0.308	0.57

Size of ore, 40-Mesh.

Sample taken after	1 Hour.	2 Hours.	3 Hours.
Zinc as normal sulphate	Per Cent. 5.82	Per Cent. 6.04	Per Cent. 5.50

Analysis of Ore at Maximum Normal Sulphate.

	Zinc.	Sulphur.	Zinc as Normal Sulphate.	Sulphur as Normal Sulphate.	Sulphur as Basic Sulphate.	Sulphur as Sulphide-Sulphur.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Actual	37.97	4.16	6.04	3.02	1.06	0.08
Corrected to 32.54 per cent. of zinc	32.54	3.57	5.18	2.59	0.906	0.07

a lower temperature. The normal, as well as the basic zinc sulphate found in these tests, contains a trace of iron sulphate.

In order to obtain another means of comparison for raw blende and roasted blende mixed with pyrite, a test was made with a 40-mesh mixture of raw blende and pyrite at 650° C. The total sulphur would not, of course, be reduced to a low

figure; that which was set free might, nevertheless, have a strongly sulphatizing effect. The maximum soluble-sulphate was obtained after 8.25 hours, 4.13 per cent. by actual and 3.49 per cent. by corrected analysis. The results of this experiment are given in Table XXIII. This is the second highest figure obtained in the sulphatizing tests with blende- and pyrite-mixtures, although the total sulphur has been reduced only to 13.8 per cent. and the sulphide-sulphur to 12 per cent. The sulphide-sulphur will have been, in all probability, in combination with zinc.

TABLE XXIII.—*Results of Roasting Blende and Pyrite.*

[Charge: 100 g. Joplin blende and 120 g. Davis pyrite; temperature, 650° C.; time, 12.75 hours. Calculated analysis: Zn, 30.14; S, 43.63; Fe, 25.43 per cent.]

Size of ore, 40-mesh.

Samples taken after.....	Hrs. 0.75	Hrs. 2.25	Hrs. 3.75	Hrs. 5.25	Hrs. 6.75	Hrs. 8.25	Hrs. 9.75	Hrs. 11.25	Hrs. 12.75
Zinc as normal sulphate.....	Per Cent. 0.68	Per Cent. 1.93	Per Cent. 3.04	Per Cent. 3.30	Per Cent. 3.65	Per Cent. 4.13	Per Cent. 4.11	Per Cent. 4.10	Per Cent. 4.06

Analysis of Ore at Maximum Normal Sulphate.

	Zinc.	Sulphur.	Zinc as Normal Sulphate.	Sulphur as Normal Sulphate.	Sulphur as Basic Sulphate.	Sulphur as Sulphide-Sulphur.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Actual.....	35.68	16.34	4.13	2.07	0.05	14.82
Corrected to 30.14 per cent. Zn.....	30.14	13.80	3.49	1.75	0.04	12.00

The ratio of sulphur as basic sulphate to sulphur as normal sulphate, is 0.026; that of the two preceding tests, 0.25 and 0.35. The enormous decrease is probably due to the complete decomposition of basic iron sulphates in the 8.25 hours, while the 3 hours of the other two roasts were not sufficient for this purpose; in none of the samples taken after 3.75 hours' roasting could any dissolved iron be detected. Between 8.25 and 12.75 hours' roasting very little soluble zinc sulphate is decomposed, the difference, 4.13 — 4.06, being only 0.07 per cent.

It ought to be stated that in all the roasts carried out at 650° C., the charge showed a tendency to rise above the given temperature as long as pyrite was burning freely. Whenever the pyrometer indicated a rise to the danger-point, the tray was removed from the muffle and allowed to cool to the desired degree of heat. After pyrite had ceased to burn, the temperature showed no tendency to rise above 650° C.

6. *Summary of Sulphatizing-Roasts.*—In order to bring together the leading features of the results obtained in the sulphatizing-roasts, the main points have been assembled in Table XXIV. The conclusions arrived at are:

1. The amount of normal zinc sulphate soluble in hot water obtained in the tests is very small. Based upon 100 parts of zinc in the charge, it reaches 9.07 per cent. with raw blende and 15.92 per cent. with dead-roasted mixed with a large excess of pure pyrite. Both extractions are too small to be of much practical value.

The fundamental difficulty in obtaining a satisfactory percentage of extraction lies in two facts: Normal zinc sulphate is rapidly decomposed at the temperature for roasting blende; at the beginning of a roast, sulphur dioxide strongly predominates over sulphur trioxide. In the process carried out in the Harz mountains, the ore is roasted very slowly in heaps which have to be turned three times; the first heap of 500 tons of ore burns from 6 to 7 months, the second fire lasts from 6 to 8 weeks, and the third from 4 to 6 weeks. Normal and basic sulphates are formed mainly in the fines, i.e., in the relatively cool cover, and these alone are leached with water and sulphuric acid. The probable explanation for the formation of these sulphates is that zinc sulphate formed toward the hotter center of a heap is decomposed, and the sulphur trioxide, dioxide and oxygen rising attack the fines in the cool cover and form normal and basic salts, or normal salts alone which are decomposed in part when the heat creeps up toward the end of the roast.

2. The ratio of sulphur as basic sulphate to sulphur as normal sulphate increases with the temperature.

3. Ferruginous (Warren) blende is more difficult to roast than blende running low in iron (Joplin), but the former gives under similar conditions of treatment a higher yield in normal sulphate owing to the presence of isomorphous sulphide, the

TABLE XXIV.—*Summary of Sulphatizing-Roasts of Blende With and Without the Addition of Pyrite.*

Kind of ore.	Joplin Blende.			Warren Blende.		a. Raw Joplin Blende and Davis Pyrite.		b. Roasted Joplin Blende and Davis Pyrite		a. Raw Joplin Blende and Davis Pyrite.
Analysis of ore.	Zn, 66.32%. S, 32.54%. Fe, 0.45%.			Zn, 53.94%. S, 31.37%. Fe, 8.80%. Pb, 3.82%.		Zn, 30.14%. S, 43.68%. Fe, 25.43%.		Zn, 32.54%. S, 31.18%. Fe, 27.43%.		Zn, 30.14%. S, 43.68%. Fe, 25.43%.
Weight of roasting charge, grams.	350	350	350	300	300	220	220	203	203	220
Size of ore, mesh to linear inch.	8	8	8	8	8	8	40	8	40	40
Temperature of roast, °C.....	690	740	790	790	900	790	790	650	650	650
Time of roast, hours.....	12.5	15	7	15	6	6.75	6.75	3	3	12.75
Maximum normal zinc sulphate, after hours.	10.5	13.5	5.5	7.5	2.25	6.75c	2.25	3	2	7.25
Weight of leaching sample, grams.	10 and 3	10	10	10	10	3	3	3	3	3
Maximum normal zinc sulphate, per cent.	0.623	2.48	1.20	4.89	0.228	2.47	2.47	2.50	5.18	3.49
Maximum basic zinc sulphate, after hours.	d.	13.5	5.5	7.5	3.75	d.	d.	d.	d.	d.
At maximum normal sulphate. — Sulphur as normal sulphate, per cent.	0.312	1.24	0.60	2.45	0.114	1.08	1.24	1.25	2.59	1.75
At maximum normal sulphate. — Sulphur as sulphide-sulphur, per cent.	15.74	1.52	0.62	5.26	8.09	1.92	0.25	0.57	0.07	12.00
At maximum basic sulphate. — Sulphur as basic sulphate, per cent.	d.	0.77	0.66	1.69d	0.45d	d.	d.	d.	d.	d.
At maximum basic sulphate. — Sulphur as sulphide-sulphur, per cent.	d.	1.52	0.62	5.26	0.61	d.	d.	d.	d.	d.
At end of roast. — Sulphur as normal sulphate, per cent.	2.94	1.02	0.40	1.86	0.034	not determined.				
At end of roast. — Sulphur as basic sulphate, per cent.	0.134	0.633	0.41	1.40	2.85					
At end of roast. — Sulphur as sulphide-sulphur, per cent.	13.65	1.37	0.19	2.35	0.16					
At end of roast. — Sulphur, total, per cent.	14.08	3.02	0.997	5.61	0.48					
Zinc extracted, at maximum normal sulphate, per cent. on zinc in ore.	0.94	3.74	1.81	9.07	4.23	7.99	8.19	7.68	15.92	11.58

a. Raw Joplin Blende, 100 g.; Davis Pyrite, 120 g.
b. Roasted Joplin Blende, 83.8 g.; Davis Pyrite, 120 g.
c. Roast not carried to maximum soluble-sulphate.
d. Maximum not reached.

basic salts of which have a strongly sulphatizing effect when they are decomposed by heat.

PART IV. APPENDIX: FORMATION OF ZINC FERRATE.

It has been proved that, in dead-roasting ferruginous blende under normal conditions, both the iron and zinc sulphides are converted into oxides. If these oxides were merely a mechanical mixture, it ought to be easy to remove all the zinc oxide by a solvent which did not attack the iron oxide, but in treating such ores it has been found that a considerable proportion of the zinc remained in the residue. Prost[32] thinks that zinc oxide and ferric oxide from zinc ferrate, $ZnFe_2O_4$, which is not readily soluble, and Ingalls also inclines to this belief.

While this question is not directly connected with the subject in hand, it seemed of interest to make a few synthetical tests in order to find out whether such a compound was formed or not. These tests consisted in heating weighed quantities of zinc oxide and ferrous sulphate, and of zinc oxide and ferric oxide, in a porcelain crucible over a Bunsen burner, and leaching the heated mixture with ammonium sesquicarbonate. A solution of this salt was chosen instead of the pure ammonia liquor, for the reason that tests made by me years ago[33] had shown that the sesquicarbonate was an excellent solvent for ignited zinc oxide, while pure ammonia was not.

The chemically pure zinc oxide used for the tests was found to have taken up carbonic acid by long standing; an analysis gave 68.48 per cent. of zinc instead of the theoretical 80.26 per cent. Crystals of green vitriol furnished the ferrous sulphate; the ferric oxide was prepared by roasting ferrous sulphate. In order to form $ZnFe_2O_4$, the proportions of Zn : Fe required are 54 : 93. The results of the series of tests made are given in Table XXV.

In making up the mixtures, the quantity of zinc was kept constant and that of the iron varied. The time of heating ranged from 1 to 2 hours; 1 hour, however, was sufficient to produce the desired result, as is seen from tests Nos. 4 and 5. Experiments in leaching proved that a time of longer duration than 2 hours gave no additional extraction, hence 2 hours was chosen as standard.

In experiment No. 1 the proportion by weight of zinc and

[32] *Bulletin de l'Association Belge des Chimistes*, x., 246–263, through Ingalls, *Metallurgy of Zinc and Cadmium*, New York, 1903, p. 32.

[33] *Zeitschrift fuer Berg-, Huetten- und Salinen- Wesen in Preussen*, 1880, xxviii., 279 and 280.

TABLE XXV.—*Results of Heating Zinc Oxide with Ferrous Sulphate and with Ferric Oxide.*

Experiment, No.	1.	2.	3.	4.	5.	6.
Proportion by weight, zinc : iron	54 : 3	54 : 9	54 : 27	54 : 93	54 : 93	54 : 9
Time of heating, hours	1	1	2	1	2	1
Time of leaching, hours	2	2	2	2	2	2
Total zinc, leached, per cent	98.19	92.96	77.85	39.42	39.65	99.91
(a) Total zinc in residue, per cent	1.81	7.04	22.15	60.58	60.35	0.09
(b) Total zinc, per cent. of, required to form $ZnFe_2O_4$	3.22	9.67	29.00	100.00	100.00	9.67
Ratio, a : b	0.56	0.73	0.76	0.60	0.60	.....

iron is 54 to 3. Of the total zinc present, 98.19 per cent. was dissolved out, leaving 1.81 per cent. in the residue. If the compound $ZnFe_2O_4$ had been formed, 3.22 per cent. of zinc, instead of 1.81 per cent., ought to have remained insoluble. With the increase of iron in the mixture the quantity of zinc held back in the residue also grows. In experiments Nos. 4 and 5, all the zinc ought to have remained in the residue, but 39.42 and 39.65 per cent. was leached out. In experiment No. 6, in which zinc oxide was mixed with ferric oxide, practically all the zinc oxide was removed by ammonium sesquicarbonate; which shows that no compound whatever was formed.

The tests, as far as they have been carried, tend to show that upon heating zinc oxide with ferrous sulphate various compounds of zinc and iron are formed, and not a single definite compound of the formula $ZnFe_2O_4$. The quantity of zinc remaining insoluble increases up to a certain point and then again decreases. Synthetical experiments with zinc oxide and ferrous sulphate cannot truly represent the process that takes place in roasting ferruginous blende; they can only point toward the direction of the reactions. The whole question of zinc ferrate must therefore be solved in a laboratory that is connected with central works treating blendic ores with different percentages of iron.

In dissolving the different samples of blende obtained in the sulphatizing-roasts, it was found that roasted Joplin blende dissolved readily in a mixture of potassium chlorate and nitric acid, but roasted Warren blende did not, but left behind a black residue, which was only slightly attacked by warm nitrohydrochloric acid, but dissolved readily when brought to a boiling-heat.

The Gold-Mines of the San Pedro District, Cerro de San Pedro, State of San Luis Potosi, Mexico.

BY GEORGE A. LAIRD, SAN LUIS POTOSI, MEXICO.

(Lake Superior Meeting, September, 1904.)

I. Introduction.

This old and once famous district played, through its enormous production of silver and gold, an important part in the history of the State of San Luis Potosi. According to a pamphlet prepared under the direction of the Governor of the State, Sr. Don Blas Escontria, and presented to the visiting members of the Institute during the Mexican meeting of 1901, this district is situated 20 km. east of the city of San Luis Potosi, in a range of mountains of moderate height, extending 135 km. N. and S., and 16 km. E. and W.; and the mineralized area hitherto developed is about 600,000 sq. m. The history of the district, as given by the record dated 1778, and formerly kept in the San Francisco convent, says that it was discovered in 1575 by Sr. Don Pedro Irarte, and adds that the city of San Luis Rey de Francisco del Potosi received its name (in allusion to the famous Potosi mines in Bolivia) by reason of the remarkable productiveness of its gold- and silver-mines in the first period of their exploration, during which they paid to the Crown, as *quintos* (the fifth part of the product), $97,000,000, as appears from the accounts of the royal treasury. The amount of the *quintos* paid during the "bonanza period" of 80 years was $72,000,000. The larger sum first mentioned probably represents a longer series of years. But either will suffice to indicate the great wealth of the mines.

In 1663, the decadence of the industry began with the caving-in of several mines, concerning which the record of 1778, already cited, says, in substance, that on Good Friday one of the owners in the enterprise, in view of the unfavorable prospect of a lawsuit in which he was engaged, set fire to the timbers supporting the underground galleries and workings of the San

Cristobal mine (in which nobody was working on that day), thus causing the caving-in of the Cristobal, Santo Domingo, San Pedro and adjacent mines.

The principal rich ore-deposits being in this ground, attempts have been made since 1760 to open an adit which would reach them. This adit, called "El Rey," was a failure, being run on a level too high for its purpose.

About 1816, the Socavon de Victoria (Victoria tunnel) was commenced; and about 1870 the present owners secured possession of the greater part of the mineralized ground, now bearing the name La Victoria y Anexas. Since that date, work has been continuously prosecuted, although this was done at a loss until 1895, when the erection of smelting-works in the city of San Luis Potosi made the enterprise profitable.

The first systematic attempt to survey and map the mines was begun in 1901 by Mr. Frederick W. Taylor, then general manager. Since then, many of the accessible workings have been surveyed by the engineers of La Compañia Metalurgica Mexicana, which operates under lease a part of the mines, and by the present administration of La Victoria y Anexas.

A topographic map of the mines in the San Pedro district is given in Fig. 1.

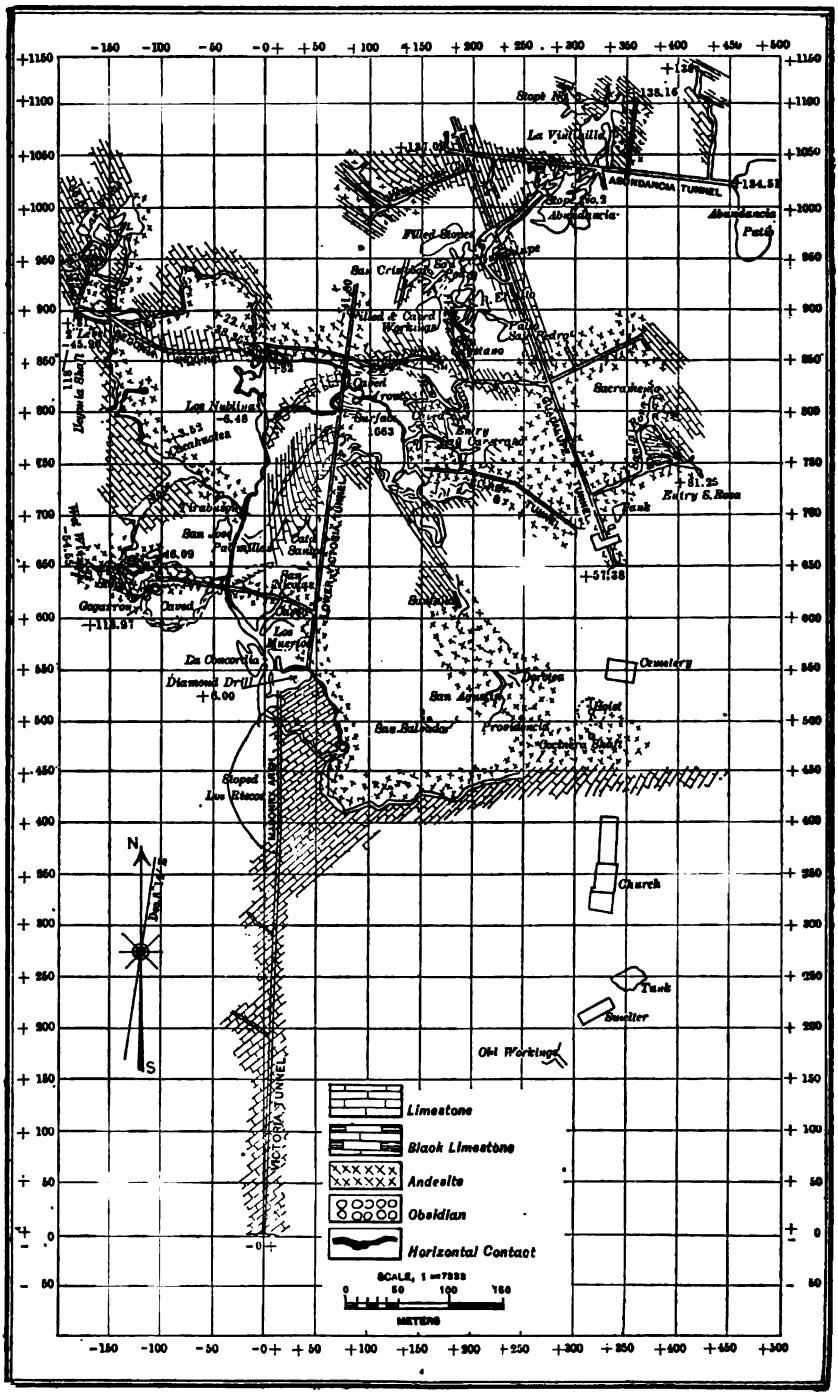
At the present time 100 openings exist—many more being lost and hidden. The work is carried on by three companies: El Barreno y Anexas, San Jose de Cocinera, and La Victoria y Anexas, the last-named owning four-fifths of the territory. The mines of the two former companies were inaccessible to me, and therefore I can speak of them only in a general way.

The territory below the level of the Victoria tunnel is under lease to the Compañia Metalurgica Mexicana.

The Guadalupe and Abundancia tunnels are operated by the Victoria company, ores being mined from the San Pedro El Alto, San Cayetano, and Guadalupe mines through the former, and from the Abundancia through the latter. This company is producing about 75 tons of ore daily.

All other mines are operated by "camperos," or short-time lessees, and are largely unsurveyed, since, by reason of frequent cavings and of promiscuous stoping and gophering, a map would be too complicated and expensive to have a value commensurate with its cost.

FIG. 1.



TOPOGRAPHIC MAP OF THE MINES IN THE SAN PEDRO DISTRICT.

II. Geology.

The geological features of the district comprise a base of andesite, overlain by limestone, which is capped on some of the highest summits by rhyolite.

The rhyolite is barren. Ores occur on the limestone-andesite contact, or in the limestone near the andesite. The latter contains small secondary veins, but never has been found to carry paying quantities of mineral, although high assay-results have frequently been obtained from specimens.

The contacts and contact ore-bodies occur at every conceivable angle, but always exhibit, when explored, a hanging-wall of limestone and a foot-wall of limestone or andesite.

The ancient workings, opened at the outcrops, followed the veins to a maximum vertical depth of 250 meters. Several tunnels were driven during the 19th century, namely: the Victoria in 1818; the Abundancia and Guadalupe in 1890. In 1903, a new tunnel, the Dorotea, was commenced, but is now suspended, by reason of questions concerning surface-rights. One shaft, the Begonia, abandoned at the depth of 250 m., somewhere between 1873 and 1885, was re-opened in 1903.

The Victoria tunnel, after having been driven 500 m., encountered the caved ground of the disaster of 1663; and a shaft was sunk 50 m., and the lower Victoria tunnel was driven 400 m., presumably with the intention of cutting the Begonia shaft; but having missed that shaft by 250 m., in consequence of a mistake in direction, was long ago abandoned.

The collar of the Victoria shaft is immediately below the limestone-contact, and all its workings are in the eruptive rock. A diamond-drill hole sunk 985 ft. below the limestone failed to cut through to any other sedimentary rock.

The mine of the Cocinera company, operated through a 250-m. shaft, is near a contact, and shows limestone at bottom. The workings on the contact show the limestone to be dipping towards the drill-hole, but I doubt its continuance in that dip to the depth necessary to bring it 500 m. below the level of the Victoria tunnel.

The contacts in the district are clean-cut; and the limestone gives no evidence of metamorphism.[1]

[1] Geology of the San Pedro District, by G. I. Finlay, Ph.D., *School of Mines Quarterly*, November, 1903, p. 60.

The extent of the andesite has not been determined, and the whole of the territory now known to be mineralized is included in the relatively small area controlled by the three companies already mentioned.

III. The Ores.

The ores, which differ widely in various parts of the territory, may be grouped in seven classes, as follows:

Table I.—*Classification of Ores.*

Class.		Gold.	Silver.	Lead.	Silica.	Lime.	Iron.	Remarks.
				Per Cent.				
1. Hematites.	A	High.	Low.	None.	.....	.....	.....	.....
	B	Low.	High.	None.	.....	.....	.....	.....
	C	High.	Low.	5 to 15	.....	.....	.....	.....
	D	Low.	High.	5 to 35	.....	.....	.....	.....
2. Carbonates.	A	High.	Low.	5	High.	High.	Low.	.....
	B	Low.	High.	25	High.	Low.	Low.	.....
3. Residual cherts.	A	High.	Low.	1 to 25	High.	.....	Low.	.....
	B	Low.	High.	1 to 25	High.	.....	Low.	.....
4. Secondary sulphides.		Low.	High.	.....	.....	.....	.....	Occurring as galena or copper and iron pyrites.
5. Secondary copper-ores.		.....	.....	.....	.....	.....	.....	Carbonates and sulphides, not in paying quantity.
6. Lime-ores.		High.	Low.	None.	Low.[a]	High.[b]	Low.[c]	[a] 25 per cent., [b] 40 per cent., [c] 3 per cent.
7. Chlorides.		Low.	High.	None.	High.	None.	Low.	In andesite.

Note.—In this classification, the terms "high" and "low" apply to values, not weights. Thus 20 g. of gold and 200 g. of silver per ton would be "high" gold and "low" silver, representing in Mexican currency about $24 gold and $7 silver. On the other hand, 6 g. of gold and 300 g. silver ($6 and $10.50 respectively) would be low gold and high silver.

It will be seen that the same class of ore is never high in both gold and silver. When the different classes occur side by side, they have entirely different characteristics, and are easily distinguished; but ores of the two extremes in gold and silver, coming from different mines, or different parts of the same mine, are often so nearly alike as to make determination, by hand-glass or sorting, impossible.

The gold occurs native; the silver, associated with the lead or as a chloride; the lead (except in the secondary sulphides), as a carbonate; and the iron as hard blue and soft red hematite. Silica appears in the ores as chert, rarely as quartz,

sometimes as agate; and lime, as the country-rock, or redeposited as calcite crystals.

IV. The Mines.

The large surface-area of the caved-in portion of the mines indicates stopes of enormous size, which are now inaccessible. Work is now going on with the intention of cutting into the caved portion in search of ores which, left as unprofitable by the ancients, would now be ranked, for modern methods of mining and smelting, as of comparatively high grade.

1. *San Pedro El Alto.*

Probably the oldest of the workings now open are those of San Pedro el Alto. Here siliceous ores occurring at the surface were followed down; and the area stoped, together with the small pockets of ore remaining, indicates that the greater part of the ancient product consisted of hematite and soft lead carbonates.

The ore-bodies must have been extremely rich and large. Free gold has been found in kidneys of from 10 to 12 kg. in weight, and ores carrying 10 kg. gold and 1.5 kg. silver per ton were extracted as late as 1902.

Two classes of ores are now being mined in the large stopes near the surface: Hematites high in gold, and low in silver and lead (class C in the foregoing table); and hard carbonates, or residual cherts, low in gold, high in silver, and carrying 25 per cent. of lead.

A hard, dark blue hematite, occurring in veins through the softer hematites, frequently carries free gold to a high value; but when it does not carry free gold which is visible to the naked eye, it is almost worthless, being practically barren of both gold or silver.

In depth, the ores occur as: Neutral, free from lead; neutral, with 25 per cent. of lead; hematites, free from lead; and siliceous ores, free from lead. They are all low in gold and high in silver. The ore-bodies almost always occur in pairs, *i.e.*, two large bodies are separated by limestone or by low-grade and unprofitable mineralized matter. The softest material, and probably also the richest of the hard ores discovered by the removal of the soft streaks, have been stoped away.

There are indications that the ancients were unable to handle the chert profitably, by reason of its hardness, although much remaining in the mines has been vitrified by the fires with which they sought to soften it. Grinding it by arrastra must also have been difficult.

The fillings of the stopes contain both fine and coarse ore, of good grade under existing treatment-conditions; and much which is now unprofitable may prove valuable at a later period.

From the surface to the Guadalupe tunnel-level (over 100 m. vertical) there are three pairs of large stopes. Below this level there are other ancient workings, which will be opened hereafter. At present any increased production from this point could not be handled.

The surface-ores now remaining are the richest, diminishing in value to the Guadalupe tunnel, where the lower workings connect and form a "U," the opposite end being worked through the Abundancia tunnel. The ore-bodies at that end are entirely different in character from the San Pedro deposits. Although about the same as to gold-content, they are lower in silver; always carry lead; are neutral as to silica and iron; and, near the surface, frequently contain as much as 15 per cent. of lime.

At Cueva Colorado, a surface-opening from the upper stope of San Pedro and about 20 m. higher than the mine-entry, ores are now produced containing about 100 g. gold; 100 g. silver per ton, and 3.5 per cent. of lead, 20 per cent. of iron and 30 per cent. of silica. The old fillings, beneath these backs, contain 30 g. gold, 160 g. silver, 5 per cent. of lead, 20 per cent. of iron, and 30 per cent. of silica.

Small stringers in the limestone hanging-wall give 6 g. gold, and 1,600 g. silver, per ton, and 70 per cent. of lead.

The northerly side of the stope contained a small 18-in. stringer of ore between limestone walls, which, being followed, led to a stope (the former entry to which has not been discovered after ten months' work), called Salon de los Mojoneras, from the fact that it contained a number of monuments of loosely piled stones.

More than 1,000 tons of ore, the greater part of which has been chert-hematite, have been removed without encountering walls.

Various small veins or feeders were followed by the ancients, one of which led to "the sick stope," so called because the miners were poisoned by arsenic. The ores carried approximately 75 g. gold and 50 g. silver per ton, and 5 per cent. of lead, and 20 per cent. of arsenic. The general mass of this ore is chert, frequently resembling quartzite, carrying the lead and arsenic in veins. The gold is free and can easily be panned out, but lead and arsenic may have made it difficult of amalgamation, and thus caused it to be left. I doubt the theory that this was done on account of the effect of the arsenic upon the workmen.

Below these stopes is another pair, in which nothing remains to denote the character of the ores extracted; but the pillars existing between the two deposits are limestone, with veins of chert, carrying 15 g. of gold, and 250 g. of silver per metric ton, and 10.4 per cent. of lead, 10 per cent. of iron and 65 per cent. of silica.

The third pair, or practically a "double" pair, contains hematites, cherts and lead carbonates. The hematites are soft and contain about 5 g. of gold and 100 g. of silver per ton, and 2 per cent. of lead, 45 per cent. of iron, 20 per cent. of silica, and 5 per cent. of lime.

The open stopes from the crest of the hill to the Guadalupe tunnel show an extraction of about 600,000 cubic meters.

At the point of union with the Abundancia workings there is a decided difference in the ores, which here carry 2 per cent. of antimony, and traces of arsenic.

The entire series of deposits are in the limestone or adjacent to the andesite, but the latter does not contain secondary veins at the point of contact, although stringers of hematite follow the line of contact for a short distance, on either side of the ore.

The general formation of the deposits in San Pedro el Alto comprises a hanging-wall of limestone, which frequently contains small kidneys of lead carbonates, carrying as much as 71 per cent. of lead, very low in gold and very high in silver. Below this, the ancients have left a hard *Guija* or residual chert, high in lead carbonates, low in gold and high in silver.

The hematites, which lie below the chert cap, are high in gold, low in lead and silver, and extend to the limestone footwall, which is practically barren.

2. *Abundancia.*

In the Abundancia no chert exists: The silica is displaced by lime. The values are about the same with the exception of the hematites on the foot-wall, which are practically barren, and large bodies of low-grade ores in contact with the andesite, which, although containing lead and having the same general appearance, are of too low a grade in gold and silver to have a mining-value. These latter bodies, constituting another "pair" of deposits, differ greatly from those of the upper San Pedro. One of them is horizontal, while the other is vertical. I do not doubt that the two are connected, although this is not absolutely certain. The vertical deposit has produced thousands of tons: 30 g. of gold and 70 g. of silver, per ton, and 10.5 per cent. of lead, 15 of iron, 50 of silica and 1 per cent. each of arsenic and antimony.

The hanging-wall is limestone, the foot-wall andesite; and although there are many small veins leading to the stope, the general character of the ores is the same, varying in iron and silica from neutral to high excess of the latter.

In the horizontal stope, which is of recent development, the ores contain, approximately, 20 g. of gold and 60 g. of silver, per ton, and 7.5 per cent. of lead, 25 of iron, 25 of silica, 8 of lime, and 1 per cent. each of antimony and arsenic. The hanging-wall is lime and the foot-wall a low-grade hematite, carrying 7 per cent. of lead, which rests on the andesite or lime.

A large boulder of limestone in the center of this stope was supposed at first to be a foot-wall, with a second deposit of hematite below. This may prove to be the case, as the carbonate ores lie on the limestone at the farthest point developed. The stope now measures approximately 40 by 20 by 10 m., with the back and one side to the country-rock. All ores are stoped to funnel-shaped mill-holes, which lead to levels driven from the main tunnel. At the farthest chute from the main tunnel, the formation is hematite capped by limestone, about 2 m. thick, which lies directly beneath the pay-ore.

From the Abundancia, a passage-way driven *pozo patilla*, or by the winze-level system (probably to keep the peons from falling more than a certain distance), has been connected with the Guadalupe and put in shape for the extraction of ores by that

tunnel. A single ore-body, from the Guadalupe level to this connection, demonstrates that veins, at least, connect all the bodies from San Pedro el Alto to Abundancia. Development-work is being carried on with the expectation of encountering the companion of this single ore-body.

3. *San Cayetano.*

San Cayetano, another of the old mines, contains three classes of ores:—Hematite, low in gold, high in silver, and high in lead; carbonates, low in gold, high in silver, and high in lead; and silicates, low in gold, high in silver, and carrying no lead.

Although the iron and the siliceous ores lie side by side, and the siliceous ores are very porous, they are comparatively little stained by the iron-ores, which in places are highly oxidized.

At the present time, the mine is being cleaned of waste and ore from below. Eight months of steady work has failed to open a passage-way from the Guadalupe to the surface. Much ore is extracted with the waste, and much remains which, owing to the loose condition of the fillings, cannot be extracted until everything is free above, which will be in six or eight months.

4. *Santo Domingo.*

The Santo Domingo is badly caved, and the ores extracted from the old fillings by *camperos* are high in gold, low in silver, high in lead and excess of insoluble material.

In these workings, occasional boulders of copper oxide are found, running as high as 50 per cent. of copper. The accompanying minerals are not discolored; and copper does not occur in paying quantities.

Here, as in San Cayetano, the effects of the cave-in of 1663 are plainly evident, many places in the passages being between large limestone boulders.

The Dorotea tunnel will cut the workings at approximately 100 m. depth; and much of the old filling can be profitably removed.

The deposits of the Abundancia, San Pedro el Alto, San Cayetano, and Santo Domingo, compose the principal district of iron-ores, with the exception of the Begonia shaft, which will be mentioned later.

These four mines may be considered as a single series of deposits, comprising ores of three classes:

1. Hematites, both hard and soft, high in gold and low in silver; or low in gold and high in silver; and low or high in lead.

2. Hard and granulated cherts, high in gold and lead and low in silver; or high in silver and lead, and low in gold.

3. Hard or soft carbonates, high in gold and low in silver; high in silica and low in iron; and high to low in lead.

5. *Other Openings.*

On the opposite side of the mountain are the openings of Cata Santos, San Nicolas, Palmillas, Los Muertos, San Pedro el Bajo, and others.

The ores of these openings are widely different from those already described, and consist of residual cherts, carrying from 5 to 25 per cent. of lead as a carbonate, and high gold with low silver, or high silver and low gold; the lead evidently carries the silver; and the ores decrease in gold with increase in lead. They run as high as 2,500 g. of silver per ton. Iron and silica remain in about the same relative proportions; and an average would be about 18 per cent. excess of insoluble. Hematites rarely occur. The lead is very fine, and not easily distinguished without panning; and the ore-bodies, as developed, do not show any contact with porphyry.

Much of the surface is badly caved in. All surface-indications in the immediate vicinity have been followed; and nearly all of them have led to ore. Although in many cases the ores thus encountered have been considered as merely superficial, I believe that they continue to the andesite, the presence of which in depth is demonstrated by the Victoria tunnel and the Begonia shaft. These workings will be developed through the new tunnel, at a depth of approximately 75 m., there being fifteen old surface openings to be passed in 250 meters.

North of these mines is the Gogarron, which carries ores of all the varieties except copper-ores found in the district. An incline, with a general northerly direction was sunk on ore its entire length. Large pillars of all grades of ore demonstrate that a large tonnage was extracted. At 169 m. in vertical depth, masses of limestone from the cave-in of 1663 are found, showing that the old workings were more than superficial.

There are passages from the lower Gogarron workings to Las Nublinas, a solitary stope, still remaining open, although all roadways are through badly caved ground. The existing conditions point to a sag in the limestone; which, although in close proximity, it lies flat, is here nearly vertical. In Las Nublinas, the ores now mined by leasers are siliceous lead carbonates, high in silver, low in gold, and carrying as high as 10 per cent. of antimony.

From Las Nublinas to the Victoria tunnel, the passage is on, or very near, the bed-rock for its entire distance, and the sides are walled with *trinchas* or piled waste rock.

The andesite is in places slightly discolored with copper carbonates and contains veins of secondary sulphides, from 20 to 100 mm. in width. Bodies of ore have been removed from above, and some of good grade still remain, but will be inaccessible until reached by new levels.

6. *Victoria Tunnel.*

From the Victoria tunnel openings were driven, large enough to permit working in all directions. Ores were encountered in many of these, which have long since been abandoned.

At the collar of the interior shaft much stoping has been done, and is still going on. Thousands of tons have been removed, and in the whole Victoria district the stopes indicate an extraction of millions of tons.

The ores now mined are highly siliceous. At the surface, where the Barreno company is working, a large excess of insoluble matter over iron is present, although from 25 to 30 per cent. of lime is not unusual. The surface-ores, high in gold and low in silver, carry practically no lead; but, in depth, the general values fall to very low gold, and high silver, and carry 5 per cent. of lead.

Traces of copper carbonate are present, and a small quantity of cuprous ores has been mined. The limestone carries secondary veins of chalcopyrite, galena, pyrite, and zinc blende, not in paying quantities.

In the lower Victoria tunnel, all of which is in andesite, there are numerous secondary veins from 20 to 100 mm. thick; and in the backs of this tunnel and the cross-cuts driven from it there are long threadlike crystals of alum. In the Pozo de

Agua, the extreme western point of the company's workings, there is a small amount of obsidian, with feldspar crystals which contain copper, native and as carbonate.

From the Victoria tunnel, the Princessa workings follow the contact, encountering a number of large stopes, which contain *almagre*, a deep red oxide of iron, used by the natives in making paints.

Lead-ores, carrying an excess of insoluble matter, also occur; and a large stope, supposed to have produced a high grade of copper-ore, shows discolorations, but no other evidences that copper really did exist there.

These workings (with the exception of the Gogarron), like those first mentioned, although not so thoroughly demonstrated by connections, are evidently on the same series of deposits, but as far as known do not touch the first group. The largest stopes in the district are in the vicinity of the Victoria tunnel. Los Riscos and La Campana are the principal ones.

At the breast of the Victoria tunnel there is a small area of black limestone, carrying secondary veins of copper, iron, lead, and zinc sulphides.

This stratum, apparently stained black by vegetable matter, occurs in the Begonia shaft also, although its continuity is not evidenced by the other mines between these two points.

7. *Begonia Shaft.*

The Begonia shaft was abandoned about 1870, at a depth of 275 m., and reopened in 1903 by La Compañia Metalurgica Mexicana, which now operates it, extracting hematites, low in gold, high in silver, and carrying 30 per cent. excess of iron over silica.

The shaft has penetrated the andesite, but cross-cuts demonstrate the limestone to be continuous to the depth of more than 200 meters.

The lowest mine-level (30 m. below the deepest known lime), driven in andesite to catch the lime, has passed by 15 m., the point expected to show the contact; and I doubt its existence at that depth. If encountered, it will not necessarily indicate that the contact extends to great depth, since all the indications above point to a general horizontal, rather than vertical, formation.

Here, as in the lower Victoria tunnel, there are no evidences of mineralization in the country-rock. All ores are found on or near the country-rock, and the general rule of lime hanging-wall and lime or andesite foot-wall holds good.

The deposits are evidently continuous from the Gogarron, similar in character, and of corresponding contents. Unlike those of the San Pedro el Alto group, they do not crop out at the surface. In my opinion, the shaft was sunk to cut the Gogarron contact.

The manway (an incline, together with *pozos y patillas*, or winzes and levels) shows evidences of much travel from below: the limestone in the hand- and foot-holds of the peons has been worn as smooth as glass.

A passage from the foot of the incline leads to San Cayetano, while one lower down leads to El Rey tunnel. This tunnel, originally driven to cut the caved ground, shows no regard for level, but is otherwise a beautiful piece of work, about 1.5 by 2.5 m. in size, and cut with gad and moil to a perfect arch in the back, with sides smooth and even. Evidently it was intended for pack-animals; but, after a few hundred meters of really fine work, the old methods were returned to, and a great deal of "hand-and-knee" work is necessary in order to get down to the Victoria tunnel level. In the lower workings, after crawling several hundred meters, one suddenly enters a level, about 50 m. long, which is as well finished as any level in the mine. How it happened to be driven is a mystery.

Much stoping has been done in the upper levels; but they are at present inaccessible.

V. Methods of Mining.

There are about 200 miles of workings, from which ores of many millions of dollars in value have been extracted, and every indication points to a future extraction of many thousands of tons of marketable ore.

How the connections between mines were made without plans, as they undoubtedly were, is unknown. Much of the work was done with the aid of fire in place of explosives. All the ore was packed to the surface by peons or burros; and, notwithstanding the low cost of labor (18 cents per day), the operating expenses must have been large.

By reason of its cost, timber was not then, and is not now extensively used. Supports were, as now, dry walls of limestone, one exception being a masonry arch 100 m. in length, which occupied 4.5 years in construction, and now sustains the fillings above the Victoria tunnel.

Stagings were built of poles lashed together with leather thongs, or with ropes made of *ixtl* (the fiber of the *lechuguilla*). That drills were sometimes used is proved by one now in the possession of Mr. H. Sturgis, of San Pedro. It is made of iron, drawn to two points on the circumference in place of a bit.

In the Begonia shaft, a *malacate de sangre*, or whim, was employed, leather *botas*, or sacks, being used in place of buckets. This has been replaced with a complete steam hoisting-plant.

Much of the ore was brought to the surface by peons, from a depth of 250 m., through passages barely large enough to admit a man (the leasers still employ peons for this purpose), and all waste was left in the mines when possible. Much of this is now worth extracting; but the extraction is rendered difficult and costly by the fact that a large amount must be handled in order to procure a comparatively small quantity of pay-stuff, and also by the lack of surveys of the stoped-out parts, many of which have been encountered in supposed virgin ground.

Much ore has been hidden by dishonest employees and leasers. I obtained more than 5,000 tons of ore from a filled stope, which the mine-foremen had declared to contain nothing but waste, and many hundreds of tons still remain, carrying less than 10 per cent. of unprofitable material.

The difficulties of working the property must be experienced in order to be appreciated. Carefully-planned exploration generally encounters "fillings," the extraction of which frequently causes complications. A 400-ft. chute, intended to reduce the cost of extraction from San Pedro, was set at an angle of 34 deg. It was known that the ore would not run at that angle, but the conditions rendered a steeper incline impracticable. Two days after the completion of the work (the bed being placed on stone-walls), the floor fell into an old stope. Everything having been repaired, a considerable quantity of ore was extracted, but in this work another old stope was cut; and as the surveys and angles of the walls showed no

probability that this was the same as the first, the work of emptying it was begun. After four weeks, the entire lower part of the chute began settling; the spreaders and braces loosened; and the emptying of the stope has been stopped until a surface-connection can be made.

The Big Cave.

Since I began this paper, the "big cave" has been penetrated. On Sept. 3, 1903, an old stope was encountered; but this is so common an occurrence, and so generally without special wider significance, that the possibility of a connection with the big cave was not considered, especially as this particular old stope was opened on the north side of the level we were driving in order to reach the cave, which we expected to find on the south side, 60 m. further on. But it turned out that we had struck a part of the big cave.

At the present time little can be said of this ground. The workings at the point of encounter contain badly-charred timbers, but it is not clear that they were in any way connected with the reported fire. The ores exposed are granulated and hard cherts on the andesite foot-wall, and carry much higher values than any of the other ores now under extraction. No hanging-wall has been exposed, but the fillings leave no doubt that it is limestone.

The cherts were evidently found to be too hard for extraction, or else not as rich as the soft iron-ores, of which small stringers remain. Partly fused rock still hangs on the walls; and ashes demonstrate that some of the chert was attacked by fire. About 10 tons of ore was found piled up, apparently for removal.

So far, our work has failed to disclose the entrance. The stope continues below the point cut by our level. The ground is not badly caved; but it is much crushed, and the air-currents through it indicate former mine-openings of more than ordinary size. Smoke from blasting rises, without entering into other parts of the mine; and our surveys give no clue to the hidden connections. An area large enough to admit 40 pairs of miners, all working on ore, has been cleared, with no indications that the entire ore-body has been exposed.

What will be found in this ground hereafter can only be con-

jectured; but it is more than probable that whatever may be found will be of lower grade than the ore which has been removed.

VI. Source and Distribution of the Ores.

The fact that the andesite has been proved, although in but one known location and that of extremely small area, to contain profitable values in silver chloride, and (in a number of cases) native silver, led me to sample the porphyry thoroughly; and on this examination I base the following conclusions:

The rhyolite is barren. No ores occur on the rhyolite-limestone contact.

All deposits at some point touch the andesite.

The andesite carries gold and silver at a general ratio of 1 g. of gold to 10 g. of silver; or 2 g. of gold to 20 g. of silver, per metric ton.

The andesite never contains visible gold or even moderately high assay-results in gold; and the ores carrying free gold (and sometimes gold crystals) never carry visible native silver, and very rarely silver chloride.

High-gold and high-silver never occur in the same ore.

It is my belief that the ore in these deposits can be directly attributed to the porphyry; and in support of this hypothesis I offer the following remarks.

The assay-returns of gold and silver from the andesite of various parts of the district were so similar, that Mr. Franklin A. Hall, the chemist at the mines, agreed with me in calling all the andesite samples "2 to 20 rock." Indeed, samples taken later from the diamond-drill core, 900 ft. below the Victoria tunnel-level, gave approximately the same contents of gold and silver. The calculations given by me in a semi-annual report to the company showed the quantities of gold and silver produced to be so nearly in the ratio of 1 to 10, that I extended the calculation to the 16,197 tons mentioned in Table II., and have since that included 20,000 tons more; these additional calculations confirming the ratio named.

That gold and silver do not exist in any known solution in the ratio of 1 to 10 is a point I shall not here discuss. Nor shall I try to explain how the precipitation from such a solution could be greater in some localities than in others; yet, the aggregate result from all localities give the ratio of 1 to 10.

In the case of deposits formed through magmatic segregation, I should expect that the mass and the magma should bear the same ratio in metallic composition, *i.e.*,—that a ton of material mechanically concentrated from the magma should bear the same ratio in metals as that concentrated by segregation.

An alteration, such as of a sulphide to an oxide, might later take place, causing an apparent enrichment, giving an increased value in gold and silver per ton, while the value per unit of volume remained the same; and a ton of pyrite or pyrrhotite from the magma showed an equal value in gold and silver as that from the segregated mass.

The San Pedro deposits present every appearance of having been formed by the action of water,—not by magmatic segregation, as I understand the term, and not by mechanical concentration; and the "missing link" is the solution which would have an equal affinity for both gold and silver, unless the deposits were formed mechanically.

The hypothesis that all the metallic contents have been leached from the magma being disproved by the remaining percentages which have caused this ratio to be noted, there can be no calculation concerning the area leached in order to produce the deposits, nor the percentage extracted by such leaching; since the badly oxidized and decomposed rock gives practically the same results as the clean rock.

Doubtless, humid analyses would have shown somewhat different results; but all these determinations having been made by fire, the average is assumed to be sufficiently correct for comparison.

Five diamond-drill samples, taken at a depth of 900 ft., and covering a vertical distance of 20 ft., show that the ratio of gold to silver varies from 1 : 26 to 2.5 : 9. The average, as shown by the results of 20 crucible assays, each charged with 15 g. of pulp, gave a ratio of 2 : 19.4 or 1 : 9.7, which I consider practically the same as 1 to 10.

All the exposed andesite is blocky, with joints or slips, which would account for the percolation of water to a considerable distance. The upper portions of the deposits may have been enriched through erosion and infiltration.

Even the hardest and firmest of the andesite slacks rapidly

upon exposure to the air; and the slips frequently contain calcite crystals at some distance from the lime.

The metallic values of consecutive shipments to the smelter, aggregating more than 16,000 tons, were calculated according to the assay-results accepted for settlements, and the ratio in grams of silver to gold for each of the various classes of ore was ascertained.

The total amount of ore, comprising about 200 lots, constituted the entire product for the period covered of the mines named in the following list:

Abundancia, San Pedro El Alto, Guadalupe (lead-ores), Guadalupe (iron-ores), Victoria (dry), Purchased from leasers (lead), (dry), total, 16,197.[374] tons.

Table II. shows the results obtained, together with the general averages.

TABLE II.—*Silver, Lead, Iron, Silica and Lime per Gram of Gold, According to Assays Accepted for Smelter's Settlements.*

Quantity.	Silver.	Lead.	Iron.	Silica.	Lime Carbonate.
Metric Tons.	Kilograms.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
950.847	0.005465	0.05018	2.5150	3.0920	0.04910
2,704.477	0.005757	0.04978	2.1920	3.0040	.....
47.801	0.014044	0.12426	2.1880	3.3770	.....
1,892.843	0.021020	0.00800	7.8220	5.4320	.....
546.589	0.008377	0.02583	1.0090	1.6290	0.02590
736.919	0.010900	0.00146	1.1320	1.9530	0.01260
121.018	0.043595	0.00261	4.5280	6.9550	0.13450
259.015	0.061000	0.00584	3.9850	8.0270	0.07470
735.921	0.013834	0.00179	1.0920	2.0640	.....
3,698.855	0.003888	0.04021	1.4887	2.0802	0.07153
2,588.805	0.016157	0.01950	1.0533	2.3803	0.01198
973.844	0.014875	0.04865	2.6387	4.0751	0.01245
776.240	0.006120	0.02012	0.8107	1.3968	0.03512
164.200	0.049022	0.04026	4.3643	4.6750	.....
16,197.[374] tons, resulting in a general average, per gram of gold, per metric ton, of 10.017 g. silver, and percentages as follows: lead, 0.3069; iron, 1.33; silica, 2.5438; and carbonate of lime, 0.31.					

The lead-contents are shown for such ores only as contained 5 per cent., which will show a slightly decreased amount; "dry" ores generally contain, approximately, 2 per cent. of lead. The table comprises only ores actually shipped to the smelter, and hence the iron-contents reported in it are probably higher than the average of the mass of the ore-deposits of this

particular class, because ores with an excess of iron, even though low in silver and gold, can be sold to smelters when siliceous ores of equal value in those metals would not pay the cost of smelting, and consequently ores too low in iron are not shipped.

The conclusion already stated, that high values in gold and silver never occur in the same ore, has been deduced from the following observations:

Andesite, contains by weight 1 gold to 100 silver (only when silver occurs as a chloride, or native, as before stated).

Hematite, contains by weight 1 gold to 2 silver;

Lead Carbonates (29.9 per cent. of lead), contains by weight 1 gold to 4,300 silver;

Chert, contains by weight 1 gold to 230 silver;

Chert-Hematite, contains by weight 10 gold to 1.6 silver.

The silver-content almost, but not quite, always exceed in weight the gold-content.

The ratio of 1 of gold to 10 of silver has been shown in but few instances by ore from a single working-breast.

Beginning with the ores lowest in gold, marketed during the period covered by Table I., individual lots have contained the quantities of metal shown in Table III.

TABLE III.—*Metallic Contents of Individual Lots of Ore.*

Gold. Quantity per Metric Ton.	Silver. Quantity per Metric Ton.	Lead.	Remarks.
Grams.	Grams.	Per Cent.	
4.0	85	1.8	Ferrous.
4.2	337	4.8	Siliceous.
6.5	628	0.0	Andesite.
8.5	53	5.0	Siliceous.
12.5	117	10.5	Siliceous.
18.0	284	5.1	Siliceous.
18.0	502	2.0	Siliceous.
26.0	334	5.6	Neutral.
28.0	58	5.1	Neutral.
78.3	182	4.4	Siliceous.
10,430.0	1,600	11.5	Chert-hematite.

It should be observed that a number of these lots came from leasers, and may have been composed of small lots of high-gold, low-silver, or high-silver, low-gold ores mixed. Smaller lots, ranging from 1 to 20 tons each (and less likely to have been mixed), have given the results shown in Table IV.

TABLE IV.—*Metallic Contents of Small Lots.*

Gold. Quantity per Metric Ton.	Silver. Quantity per Metric Ton.	Lead.	Gold. Quantity per Metric Ton.	Silver. Quantity per Metric Ton.	Lead.
Grams.	Grams.	Per Cent.	Grams.	Grams.	Per Cent.
2.0	1,538	25.5	50.0	1,100	7.1
6.0	25,800	29.9	60.0	120	11.3
10.0	3,943	21.4	98.0	69	4.0
12.3	714	20.7	100.0	101	57.7
30.0	30	6.0	100.0	2,140	2.0
35.0	45	7.4	103.0	247	37.7
35.0	55	9.6	120.0	204	1.1
35.0	16,000	41.6	164.0	176	1.2
46.0	64	7.6	286.0	160	9.2
48.5	2,825	14.6	522.0	232	....
49.0	417	2.8			

It will be seen that the high gold-results are accompanied by low silver, and the high silver by high lead and relatively low-gold. The highest gold-values were contained in a chert-hematite, that is interstratified chert and hematite.

Analysis of the ores shows them to range from 50 per cent. excess of iron to 80 per cent. excess of silica. A large part of the present production consists of ores carrying an excess of insoluble matter. As to the metallic value of each class of ore, I have found no general rule, with one exception, namely, that the high lead carbonates occurring in the limestone hanging-wall always carry high values in silver.

In the majority of cases, high-lead is accompanied with high-silver; yet Abundancia mine-samples have given the results shown in Table V.

TABLE V.—*Assays of Abundancia Samples.*

Gold. Quantity per Metric Ton.	Silver. Quantity per Metric Ton.	Lead.
Grams.	Grams.	Per Cent.
10	100	22.0
20	80	19.0
31	89	17.5
34	76	24.0
49	79	22.5

On the other hand, the andesite-ores, as well as others high in silver, carry no lead.

A Geological Cross-Section of the Western Cordillera Along the Rio Huasco.

BY SYDNEY H. LORAM, CANUTILLO, HUASCO, CHILE, SO. AMER.

(Lake Superior Meeting, September, 1904.)

THIS paper, which is merely an arrangement of data collected during several hurried journeys, is offered to serve as a record, until such time as a better substitute be compiled.

My observations were confined, as far as possible, over the entire distance, to the actual river valley; because near the Central Cordillera the country is so mountainous as to be impassable in any direction but that of the dominant valleys. The river is not navigable anywhere, and for the first part of its course is a mountain torrent that runs down the bottom of narrow, almost vertical-sided valleys, through which the only road requires repeated fording, and often following up the bed of the stream; but the rock-walls, standing clean and free from débris on either side, present the different formations with remarkable distinctness. A topographical map and a cross-section of the Rio Huasco is given in Fig. 1.

The Rio Huasco practically rises from *Laguna Grande*, a lake about 17 km. from the Argentine boundary, and 3,135 m. above sea-level, which is fed by direct snow-water, and by a few streams varying from a considerable volume, when supplied by the melting snow, in late spring, to little or nothing during the rest of the year. It is a somewhat pear-shaped body of water, about 1.5 miles in length and half a mile in average width, with a reported maximum depth of 45 meters. It was formed by the choking up of a narrow continuation of the same valley with blocks of an agglomerate mentioned later.

The present level is some 5 m. below high-water mark (to which point, according to local accounts, it has not been filled for 35 years). In the intervening margin is a calcareous deposit about 4 m. thick, formed by the rapid evaporation of the water in the exceedingly dry air, and aided by the capillary action of an aquatic plant, growing in thick masses around the edge of the lake. These masses, dying and banking up, form

a sort of wick, up which the water passes and evaporates, leaving behind its solid contents, which, together with the product of the decomposition of the plant itself, constitute an almost perfectly white deposit.

Around the lake for a few miles (as far east as my knowledge of the country goes) is an agglomerate of general dark-red color, varying from sintery structure (which breaks along the joints), in the upper portions, to a solid fused mass near the aplite upon which it rests. In general, the pieces of included rock are small, less than 0.5 in., and not more than 3 in., in diameter. They are fragments of dark-green aphanite, fine-grained gneiss, eurite, white feldspar (andesine) and some crystals of dark-green hornblende, varying according to fusibility from sharply angular forms to those that are rounded. Epidote crystals in bunches, formed at the expense of the hornblende, are abundant. The ground-mass or cementing material is dark red, and contains, besides microscopic fragments of the above rocks, porphyritic crystals of white feldspar, developed from itself. Throughout the mass there is a decided stratification, with a general dip of about 15° West, which, together with the fact that bending of the constituent particles (so common in the neighboring eurites) is entirely absent, makes it doubtful whether it originated as a lava which flowed over the granite, or whether it existed previously as a sedimentary or talus, and was afterwards fused in place by the granite from underneath. The appearance, within a short distance on either side of the contact, of a mixture of the two materials, and, further away, of included portions of the agglomerate, strongly favors the latter theory. The thickness of the agglomerate is unknown, but on the assumption of an average inclination of the surface of the underlying granite, it is about 750 m. where least removed by denudation. Resting unconformably upon the agglomerate are beds somewhat similar in structure, except that they consist of fragments of granite in all stages of disintegration and decomposition, and have not been subjected to heating. In general, these beds are of a light-brown mud-color, and show distinct horizontal lines of stratification. Though readily disintegrated by a blow from a hammer, the material has sufficient coherency to stand practically vertical in exposures of the entire thickness of about 300 m. Probably the

origin of these beds is the disintegration (under the violent extremes of heat and cold to which they are subjected) of the peaks further east, which may form the actual divide,—but this is only supposition, since the locality was not visited. About 3 km. west of the lagoon, and 400 m. lower down, in a gorge cut by the river, the contact between the agglomerate and the granite is clearly shown. Here, as already remarked, pieces of the former may be seen included in the latter.

The granite consists principally of pink orthoclase and quartz, with some oligoclase, very little hornblende, and small scales of muscovite showing only at rare intervals. The general mass might almost be termed aplite. The general texture is coarse; and the color, throughout its extent of 70 km. westward, varies from all but white to a decided pink. About midway across, the river has cut a small cañon about 70 m. deep, the walls of which show several intrusions (principally along the joint-planes) of dark-green aphanite.

Near its western edge, the aplite merges gradually into gneiss, dipping sharply to the west, and consisting of anorthite, hornblende and biotite. Hereabout timber-line is reached. Above, is snow- and wind-swept sierra, devoid of vegetation, except in a few spots sheltering a little poor grass and a shrub on which the vicuñas manage to exist; below, the descent becomes more gradual, and the narrow valley is cultivated.

Upon the gneiss lies limestone, with an approximate visible thickness of 250 m., dipping about 45° W., and showing its stratification very distinctly, in almost unbroken planes. It has been subjected to considerable heating; and the dark red of the upper and lower beds shades through yellow towards the blue-gray center of the mass. Fossils are so thickly distributed in the upper strata as to form a considerable proportion of the entire mass, but in the lower layers, though the same varieties (and no others, so far as I have been able to find) still occur, they are fewer. In order of numerical importance they are: *Pentamerus*, resembling *galeatus rhynchonetta* (three species, one *rhynchonella varians*, two unidentified), *spirifer* (*Walcottii* and *Galber*), *Terebratula*, and very rare (one only found) *Pecten islandicus* (Chlamys). It seems probable that this is part of the formation which contains the coal-deposits of Maricunga, about 220 km. to the north. These deposits are of considera-

ble extent, but, so far as has been proved hitherto, they are commercially without value, on account of the poor quality of the coal. It must be inferred that the aplite upon which the limestone primarily rests is post-Carboniferous. It has been reported that, along the contact of the gneiss and limestone, there are deposits of gold in the gneiss, and of silver and lead in the limestone; but no such mines are now worked; and this statement is made upon hearsay only.

Resting upon the limestone there is a small layer of dark-red agglomerate, similar in every respect to that before mentioned on p. 880.

Upon this agglomerate there is eurite, here of a general brown-pink color, and usually showing porphyritic crystals of bytownite. It forms a mountain mass extending about 6 km. to the west.

Here diorite makes its first appearance, being exposed by the action of the river, as an eruptive traversing the eurite. It is very coarse-grained, and shows dark-green hornblende in almost equal proportion to the feldspar; but it is soft, and more or less altered to a considerable depth, and if samples could be taken below that depth, it is probable that the hornblende would prove to have been originally augite. Quartz is absent; and the feldspars are labradorite and anorthite, beginning to pass into a saussuritic condition.

After passing through the diorite at Marquexas, mica-schist is encountered, forming both sides of the valley, and rising at a very steep angle to the height, in some places, of 800 m. above the river. It dips east about 50°; and, lying unconformably on its upturned edges, there are patches of horizontally stratified sandstone, which, as the sole portions of this stratum which have escaped denudation, form light-yellow flat caps to some of the summits.

The mica-schist rests on diorite, which scarcely does more than reach the surface, and would escape observation, except where the valley cuts through it.

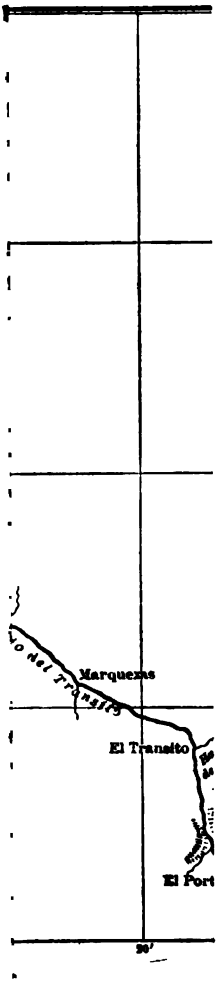
West of the diorite, hornblende-schist is encountered, extending about 4 km., and dipping sharply west. Resting conformably on this schist, pyromeride comes, consisting of a dark-red ground-mass full of quartz spherulites, most of which are hollow with their crystal-points turned inwards. Some are

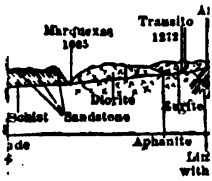
solid quartz and others have a nucleus which appears to be fine-grained granite. In these latter cases, crystals of magnetite run through the envelope of quartz. The ground-mass (which contains porphyritic crystals of anorthite) is much cracked and fissured, apparently as the accompaniment of cooling. This formation is only a few meters thick, and to the west it is in contact with aphanite which has probably caused its origin. For a short distance from the pyromeride, the aphanite contains large numbers of pieces of included rock fused into it, and in cases partially dissolved by it. Further away from the contact these included pieces become rarer, until they disappear altogether, leaving an ordinary dark-green aphanite, which extends with little change of general appearance for about 9 km. Here limestone again occurs, dipping 22° E. Its upturned western edge is exposed to the full height of about 300 m. vertical. It rests on aphanite; and, at its lower or western contact, the limestone has been converted into marble, which is nearly white close to the aphanite, changing to dark-green, and almost black, higher up; above this is yellow limestone only partially altered. Quarrying was done here, at one time, to a limited extent; and the quality of the marble is said to be excellent. Naturally, few fossils have survived the heating undergone by the rock, and these have been so fused that it is impossible to separate them from the surrounding stone. One, broken in sections, appeared to be a large *Bourgelia;* the remainder were unrecognizable. Along this contact, to the south of the river, are the once famous silver-mines of "Agua-Amarga," "Tunas," and "Viscachas," which probably correspond to the districts of "Chanarcillo" and "Tres Puntas," about 200 km. to the north. In the former of these (according to Mr. Grey)[1]. Carboniferous limestone and calciferous bituminous strata were cut to the extent of 410 ft. The reported existence of ammonites (which I have not had the opportunity of examining) in the vicinity of the above-mentioned mines, may indicate that strata of a later date also exist at these points. In general, this would seem to point to both the outcrops of limestone in the present section as being of common origin and forming sides of a syncline. The aphanite at the immediate

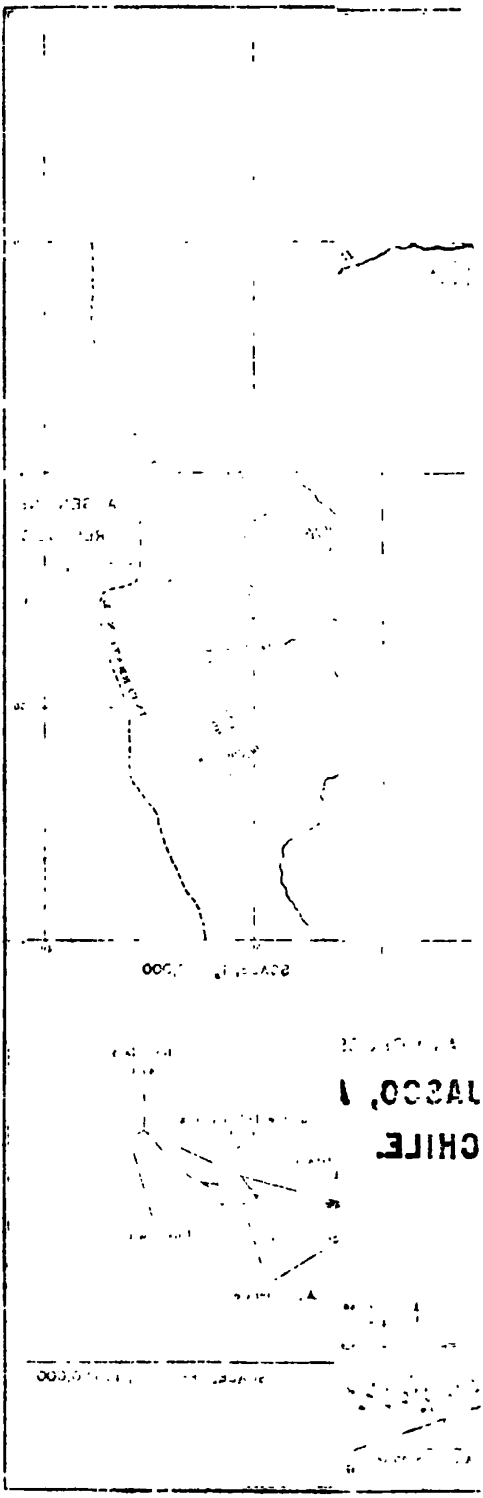
[1] *A Treatise on Ore-Deposits*, by J. A. Philips and H. Louis.

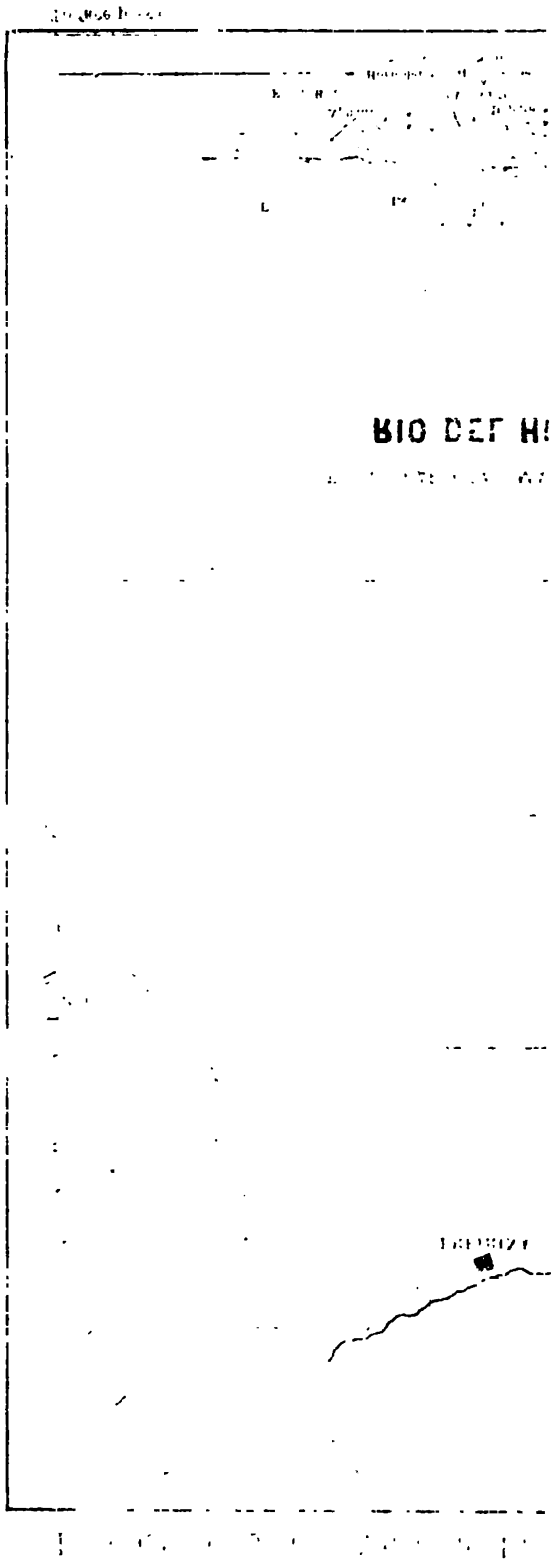
contact to the west of the limestone contains large numbers of rounded stones which have the appearance of being water-worn. They are difficult to determine, owing to the temperature to which they have been subjected, but appear to have been syenite. As might be expected, considerable quantities of epidote have been developed there; and these encrust the feldspar in such a way as to look at first sight like considerable blocks of pure epidote. About 4 km. due west there is another deposit of limestone, dipping parallel to the last-mentioned, but only about 10 m. thick. At its upper or eastern contact is a very basic agglomerate, apparently formed from a mixture of hot aphanite with pieces of limestone, in which the feldspar has been only slightly affected, while the hornblende has gone into combination with part of the limestone, forming glass (tachylyte). The balance remaining as eroded granules of marble.

Five kilometers further west the aphanite has, for a short distance, a dark sintery appearance, shows a decided purple hue on weathering, and contains porphyritic crystals of anorthite, in many of which clusters of hornblende crystals are developing. From this point the valley is filled with deposits of water-worn pebbles, the greater part of which are about 5 in. in diameter, with enough fine material to fill the spaces between them. The general dip of their longer axes is about 10° W. The average thickness of this deposit, along the center of the valley, is about 100 m., and, its width depending on the configuration of the valley, varies from a few hundred meters to about 9 miles. The surface is broken into steps or benches parallel to the river, of which five are generally visible, the first being by far the widest, and the last two often not more than from 10 to 20 m. wide. The river has cut its way through them and runs near bed-rock more or less down the center. The surfaces of these deposits are perfectly flat, with a slight inclination to the sea. They form, when irrigated by canals from the river, valuable farming-land, which is the basis of the principal industry of the valley. No fossil remains are to be found; but this is a natural consequence of the decomposing action of the percolating water and the air, which is so strong that diorite pebbles 3 in. in diameter are sometimes so decomposed that they can be cut through with a knife, leaving a polished surface, somewhat like wax, in which the remains of the original constituents are


Marquexas
El Transito
El Port


Marquexas
1903
Transito
1812
Diorite
Basalt
Sandstone
Aphanite





easily recognized. The pebbles are of the same composition as the various rocks that form the valley, diorite and aphanite predominating. According to Darwin, they were originally estuary gravels, formed during the gradual elevation of the land. Close to the sea, the valley becomes narrower, and the bottom-land, occupied by the river when in flood, widens until, at the coast, the gravel-deposits almost disappear. Above point No. 10 (see Fig. 1) a small deposit exists for a few miles in intermittent patches, and gradually disappears. From the above-mentioned point the line of observation runs along the southern edge of the gravel-deposits. Aphanite continues in its usual dark-green form until, just at the town of Vallenar, a spur of syenite crosses the valley. To the southeast, this rapidly opens out, forming a pear-shaped body. At its contact with the aphanite a little copper is found, and further away in the same direction, close to a contact with limestone, cobalt is found in conjunction with a little nickel and silver, as arsenides with iron. The syenite is in general fine-grained, and rather low in silica (52.42 per cent.) by reason of the entire absence of quartz, and the somewhat large amount of hornblende present, which latter may have existed originally as pyroxene. Orthoclase in clusters of small pink crystals forms the bulk of the feldspar, but oligoclase exists in considerable quantities, together with rather widely scattered porphyritic crystals of anorthite.

To the west of the syenite the aphanite slowly changes into diorite, until, just west of Bodeguillas, the valley is crossed by another spur of syenite, which has been considerably changed by its contact with the diorite and consequent heating. Around the contact a little copper is mined. The gravel-deposits hereabout contain large quantities of gypsum, the crystals of which cement the pebbles together. A few miles north of the river a deposit of gypsum is worked commercially.

Further west, the diorite continues for some distance; it consists of labradorite, some anorthite and augite, which is often altering (or already altered) to hornblende, and probably, with the intervening aphanite, forms part of the same eruption as that met with above Marquexas, but having been under the water of the Pacific until a more recent date, it shows fewer signs of decomposition. Around its extremities, near the contact with the neighboring rocks, as, for instance, at Canutillo

to the south and Capote to the north, gold is found. Copper also occurs, but generally in quantity too small for profitable working. In the neighborhood of these deposits, the diorite contains some quartz; but it is my present opinion that this is of secondary origin, and does not belong naturally to the mass of the diorite. At the town of Freirina, and for some kilometers west, the country consists of aphanite and hornblende-schist, often so intermixed that in a space of 5 m. square, several samples might be taken representing definitely each of these rocks. At the western extremity of this formation (at Las Tablas), pure hornblende-schist is found, dipping sharply east; beyond, to the sea-coast, is aphanite, which, on nearing the Pacific, shows every sign of its once molten state. Resting on it at the shore, and just behind the Port of Huasco, as mentioned in the paper, Notes on the Gold-District of Canutillo, Chile,[2] are to be found the fused and baked remains of shells ranging in age from the Upper Cretaceous to recent date.

In conclusion, thanks are due to Mr. Ernesto Frick, late of the Chilian-Argentine Boundary Commission, for the loan of maps of the little-known interior, from which the eastern portion of the accompanying map was constructed.

The Taviche Mining-District near Ocotlan, State of Oaxaca, Mexico.

BY H. M. CHANCE, PHILADELPHIA, PA.

(Lake Superior Meeting, September, 1904.)

THE Taviche district is about 12 miles from the town of Ocotlan in the State of Oaxaca, and about 250 miles south of the City of Mexico. Its altitude rises to something more than 5,000 ft.; and although within the tropics (latitude about 17° N.) the climate is mild and salubrious, neither excessively hot in summer nor cold in winter. The phrase, "perpetual spring," which has been applied to it, aptly describes the weather during my stay. Sugar-plantations, bananas and other tropical and semitropical fruits flourish in the valleys, while the hills

[2] See p. 696.

and mountains, from 1,000 to 3,000 ft. higher, are covered with corn-fields and vegetable-gardens.

The poorer natives usually live in huts and houses of cane or reed, roughly thatched with the same materials, or with grass or palm-leaves. Groups of these huts much resemble the villages of central Africa. The people are unambitious, and, though fairly industrious, wretchedly poor. But they are generally good-natured, contented and cheerful, and enjoy life; so that their lot is not so altogether deplorable as might at first be supposed.

Wages are very low. Miners and tolerably good mechanics earn about $1 Mex. per day (about 40 cents gold). Laborers (peons) about mines, industrial works, and plantations are paid from 40 to 60 cents Mex., say from 16 to 24 cents gold—per day. On plantations relatively remote from the railways, wages are still lower.

At one of the mines, I learned that the housekeeper who was also cook (and a very good cook too), chambermaid, laundress and seamstress for a family of four, received $2 Mex. (80 cents gold) per week, and her assistant one-half as much.

All native products, such as vegetables, poultry, grain, etc., are very cheap. In the market-place at Ocotlan I saw fine turkeys sold for 50 cents Mex. (less than 20 cents gold) a-piece.

The history of the Taviche mining-district goes back to prehistoric times. The mines were worked by the Aztecs or Toltecs, and possibly by their unknown predecessors. Yet, notwithstanding their antiquity, they are not deep; and the most extensive workings would represent what, by modern American methods, would be accomplished in a comparatively brief period. The historic records of the mining carried on at intervals for the past 300 years enable us to realize dimly the excessive slowness and inefficiency of the means employed. The older work was undoubtedly done by the firing-method. A wood-fire was kept burning against the face of ore to be extracted until, for a few inches in depth, the ore became cracked by the heat; or, this failing, water was thrown against the heated surface, and a few pounds of ore were thus cracked and broken loose. After the workings had reached a considerable depth, the heat and smoke prevented the miners from maintaining more than a single fire at a time. It is almost incom-

prehensible to the modern miner how anyone could exercise such patience and perform such labor to win the paltry daily sum which was the miner's reward. Yet one can walk along the outcrops for hundreds of feet and see the walls of the vein still standing, from which the ancients, with infinite pains and perseverance, extracted the ore by this method.

The veins of the Taviche district, usually ranging from 3 to 12 ft. thick, are generally highly inclined, although at some mines the dip is gentle.

The country, as a rule, consists of eruptive rocks; and no evidences of sedimentary strata appear near the mines.

The ancient methods of working were followed, to the almost complete exclusion of modern ones, until about 20 years ago, when a number of mines were equipped with improved appliances, and the district has become, in consequence, fairly productive, especially within the last 5 or 6 years, during which period several mines have records of shipping from $1,000,000 to $5,000,000 Mex. worth of ore. This work has been carried on principally by Mexicans, although a few Germans and Americans worked in the district. Within the last year, however, Americans have purchased nearly all the producing mines (about a dozen), have equipped some of them with modern machinery, and are beginning to extract ore on a scale hitherto unknown in the district.

Few of the mines have encountered much water; and it has been possible to extend the work by drainage-adits to depths of 300 or 400 ft.; but a deeper development will require pumping-plants. In recent years most of the ore has been raised by large horse-whims (called "malecates"); and where much water had to be raised, these malecates have been built with three or four arms, so that six or eight mules or horses could be used in working them.

Prior to the installation of whims, the ore was carried out in baskets on the backs of peons, and this method is still in use at a number of mines. At the San Francisco, peons carry ores and water up the inclined passage to the surface. Their wages are about 50 cents Mex. (20 cents gold) per day for this work.

Mexicans are noted the world over as expert ore-sorters, and to this work the boys and old men naturally apply themselves,

as in the Pennsylvania anthracite regions, boys and old men become slate-pickers in the coal-breakers. The Mexican ore-sorters earn from 30 to 60 cents Mex. (from 12 to 24 cents gold) per day, depending upon their industry and ability, the work being generally done at contract- or "piece"-rates. Notwithstanding these wages, however, the cost of ore-sorting is not small, because, in order to raise the ore to a high shipping-grade, and at the same time to reject only rock of little value, it is necessary to break it to a small size, examining each piece and breaking and rebreaking it until the pieces finally selected consist solely of ore of high grade on the one hand, and waste material of very low grade on the other. Hence final sorted ore contains few pieces larger than 1.5 in. in diameter, and ranges from that size down to very fine material.

To illustrate the amount of labor involved by the ore-sorting process: At one mine the cost of sorting was about $15 Mex. per ton, the sorters earning about 40 cents Mex. per day. In other words, 37 days' labor required to sort out 1 ton of shipping-ore. This condition existed at a mine when the value of the crude ore averaged about $36 gold; that of the rejected waste, about $12 gold; and that of the shipping-ore, about $180 gold, per ton. About 7 tons of crude ore was required to furnish a ton of shipping-ore.

With modern milling-facilities, the waste rejected at such mines should be treated at good profit; but the mines have heretofore depended upon the shipment of high-grade ore to the smelters, and have been satisfied to allow the low-grade material to accumulate on the dumps.

The transportation of ore, mining-machinery and supplies is done in rough bull-carts or by packing on mules or burros. Oxen are used for most of the heavy hauling, and nearly all of the plantation-work. The yoke used is the typical Mexican yoke, which rests upon the back of the head in contact with the horns, and is securely held in place by a heavy leather or raw-hide strap wound around each horn and around the yoke a sufficient number of times to bind the head immovably to it. Every movement of one animal, however slight, moves the head of the other, and the team is thus controlled far better than with the style of neck-yoke used in the United States. The device is effective in controlling the animals, but is rather

cruel, as it forces them to hold their heads in a strained position. It is, moreover, not efficient in developing the full tractive-force, because the animal pulls the load by pushing with the head instead of the shoulder. The Mexicans take excellent care of their oxen; feed them well; never overwork them; and thus keep them fat and in fine condition;—whereas their work-horses, mules and burros are neglected and misused, and generally are a sad-looking lot of small and weak animals.

Two average American horses exceed in draught-power four or five Mexican horses or mules. The practice in this part of Mexico is to harness five animals to an ordinary two-seated carriage, two as pole-horses and three as leaders. Two drivers are required for such a team—one incessantly plying the whip, the other handling the reins and assisting in urging the team by a continuous hissing, like that of the escape of steam from a leaky pipe. With these auxiliaries such a team is able to draw over fairly-level roads two persons and the drivers at a fair rate of speed; but upon approaching a hill, the passengers and whip-plier usually find it necessary to dismount and walk; and when very steep places are reached it is necessary to assist the team by pushing the wagon.

Horses and mules reserved especially for riding are better fed, larger and of better grade. Hence traveling is done almost entirely with saddle-animals, even in those sections where roads exist which are kept in fair condition for the ox-carts used in freighting.

The mining-methods used in the Taviche district are similar in every respect to those used elsewhere throughout Mexico. The Mexican miner invariably attacks the ore from above, and constantly follows it down, always working on top of the ore to be removed, and moving his point of attack from place to place on the vein, always following down the best ore, unless by accident he finds a body of good ore above his workings which cannot be extracted except by approaching it from below. Hence in a Mexican mine one never expects to see any quantity of ore of good grade "in sight" except in the bottom, and more especially in the lowest and deepest holes excavated below the general level of the bottom. Of course, if the mine makes any water these holes are flooded, except when active operations are going on. The mining expert, therefore, must base his valuation largely upon circumstantial evidence, in-

cluding the character of the dumps, the past history of the mine, the value and character of low-grade ore not extracted by the Mexicans, the assay-value of tailings, the general characteristics of the deposits, etc.; and he is seldom able to base an estimate of value upon ore actually "in sight."

In other words, the Mexican never blocks out ore-reserves, but, on the contrary, extracts everything valuable as fast as it is brought to sight. Of course, under this management, regularity of output is impossible; the mine at times producing nothing, and at other times is in "bonanza," with every miner breaking ore.

The history of the Taviche mines is thus, like that of most other Mexican mines, one of alternate activity and idleness. Under the Mexican system, as already described, all the high-grade ore in sight is gouged out, and either the work then stops, the mine is abandoned and the title lapses, or else the title may be kept good by permitting poor miners to continue the work as lessees, in which case, perhaps, a lessee accidentally discovers the continuation of the ore-chute, or a new chute, whereupon general operations are resumed, the good ore is gouged out as before, and regular mining is again abandoned, when, as in many cases, the title is allowed to lapse. It often happens, after a long period, that the mine is again "denounced" (located), and that the new owners, sinking to greater depth, open a new chute or pocket of high-grade ore. Such is the history of most of the old mines of the Taviche district.

The Benjamine mine has a recorded production of several million dollars' worth of ore; and its extension, the Escuadra, is said to have yielded $2,000,000 in the 5 or 6 years preceding its purchase by the present American owners. All the mines on this belt, comprising three or four distinct lodes, yield ore similar to that of the Tonopah district in Nevada.[1] The

[1] Early in 1903, I made my fourth visit to the now well-known Tonopah mining-district in western Nevada. The story of the discovery of the original Tonopah mine is a typical western romance; and the rapid growth of the town to a population exceeding three thousand, within little more than a year after the first ore was shipped, characterizes the energy with which such discoveries are exploited in the United States. The development of the camp has progressed so rapidly during the last year that I would not now venture to describe it. Company after company has been organized, until 20 or 30 have been engaged in shaft-sinking and other development-work; and in some of the new shafts, sunk where there were no surface-indications of ore, good ore has been found.

resemblance is so remarkable that in many cases an ore-sorter could not detect any difference.

The Taviche ores like the Tonopah ores are siliceous, consisting principally of quartz carrying gold (often free) and silver; the latter occurring near the surface as chloride (horn-silver), but at depths of from 200 to 400 ft., principally as black sulphide and as ruby-silver. The ratio of the metals by weight is about 1 to 100, that is, there is about 1 oz. of gold present to every 100 oz. of silver,—or in money values, about $20 in gold to $50 in silver.

Within a radius of 3 or 4 miles from the Escuadra mine many other ore-bodies have been found, some of which have yielded large returns, among which are the Providentia (which is a modern mine), Zapote, Carpenter, Conejo Colorado, Conejo Blanco (an enormous old open-cut working), Trinidad, California King, San Carlos, Indiana, Altoona, and the Chivo and Oaxaca. Most of these mines have been bought by Americans within the last two years; and since my visit, doubtless many others also have been purchased.

The State of Oaxaca seems destined to become popular among mining-investors; its fine climate, great variety of agricultural products, fruits and vegetables, enable one to live comfortably and cheaply, and insures the continuance of an abundant supply of cheap and relatively good labor, since labor is always best and cheapest where the laborer is able to live comfortably at small expense.

IN MEMORY OF
HENRY CORT
BORN AT LANCASTER 1740
INTERRED AT HAMPSTEAD 1800
TO WHOM THE WORLD IS INDEBTED
FOR THE ARTS OF REFINING IRON
BY PUDDLING WITH MINERAL
COAL AND OF ROLLING METALS
IN GROOVED ROLLS.

The Case of Henry Cort.

BY CHARLES H. MORGAN, WORCESTER, MASS.

(Lake Superior Meeting, September, 1904.)

THE case of Henry Cort comprises: (I) the nature of Cort's inventions; (II) their value to England and to mankind; (III) the remuneration received therefor by him or his family; and (IV) the suitable permanent record and recognition of his services by the representatives of the art he founded—a debt long overdue, and still unpaid.

I. CORT'S INVENTIONS.

Under this head, it is unnecessary here to enter into details. The facts have been repeatedly published,[1] and a brief outline of them will be sufficient for my present purpose.

Henry Cort, born in 1740 at Lancaster, England, became a navy broker in London about 1765, and gained from that business in about ten years something more than £20,000, which he devoted to perfecting the manufacture of iron, building a mill at Fontley, near Portsmouth, and prosecuting, between 1777 and 1783, the experiments which culminated in his two patents, one for puddling iron and the other for shaping it by rolling between grooved rolls. Careful study of the contemporary evidence, including Cort's specifications, the testimony (in act as well as word) of his business rivals, the public declarations of eminent authorities, and the history of the times, leaves no possible doubt that Cort was the real inventor of

[1] See Cort's British patents, Nos. 1351 (of 1783) and 1420 (of 1784); also, a series of articles by Thomas Webster, an eminent authority on Patent Law, in the *Mechanics' Magazine*, vol. 2, London, 1859; Percy's *Metallurgy*, "Iron and Steel," London, 1864, pp. 627–639; *Memoirs of Distinguished Men of Science*, London, 1864, p. 152; Smiles's *Industrial Biography*, London, 1897, p. 114; Facts and Proofs Collected by R. Cort, London, 1855; and a partial summary, in my Presidential Address, "Some Landmarks in the History of the Rolling-Mill," delivered before the American Society of Mechanical Engineers, at the New York meeting in December, 1900, and published in vol. xxii. of the *Transactions* of that Society.

these two inestimable improvements. Every such invention is subject to claims of priority advanced in behalf of unsuccessful predecessors. The relative rights of such claimants to the sentimental credit, so to speak, of a given step of progress, may be difficult of adjudication; and sometimes the suggester of an idea which he never effectively executed may deserve praise and thanks for his incomplete achievement; but the principles of both English and American patent law make short work with such pretensions. That law was not instituted to reward prophetic genius or intuition. It rests upon the proposition that a man who has discovered and successfully practiced an improvement in the arts, and who might possibly keep it as a trade-secret, to die with him, shall be induced, by the grant of a monopoly for a limited term, to tell his secret completely to the public, so that, after the expiration of that term, any expert in the art concerned may be not only entitled but enabled to practice it. It is a question, not of recognizing meritorious aspiration or endeavor, but of buying for public use an actually successful device. Much credit is often fairly due and freely given to unsuccessful inventors who have, nevertheless, no rightful claim to the reward of a patent-monopoly.

In the case of Cort, Dr. Percy mentions one or two prior inventors,[2] discovered by his researches, who seem to have been working on the same lines, and to some extent with the same general ideas. But they do not seem to have put their ideas into practical use. The malicious and false testimony, denying the practical merit, and thus the patentable character of his inventions, will be briefly alluded to later. It does not deserve serious consideration here. On the whole, it may fairly be said, that beyond all honest doubt and by substantially universal acclaim, Henry Cort was the first to perfect, put into successful operation, give to the world by sufficient description, and teach to other operators, his licensees, the puddling of iron and the rolling of puddled iron between grooved rolls.

[2] As to puddling, Thomas and George Cranage (British patent, No. 851, of 1766), and Peter Onions (No. 1,370, of May 1783). Since Cort began experimenting in 1777, the date of the latter is not proof of priority of invention. There is no evidence that either of them was commercially practiced. Concerning the use of grooved rolls, I think Cort's originality is not denied.

II. The Value of Cort's Inventions.

As I have already observed, Henry Cort accumulated a fortune of more than £20,000 as a navy broker, furnishing supplies for the British navy, at a period when England stood at bay, fighting for her colonial possessions in North America, and almost for her own national existence, and when the power and prowess of her navy was her chief, if not her only, reliance. Yet no forge or furnace in England could make iron fit for navy use. The nation, though endowed with vast deposits of iron-ore and of coal, was dependent upon the purer raw materials of Sweden and Russia for wrought-iron of good quality; and the Admiralty specifications called for Swedish or Russian iron. While Cort was in business as a navy broker, the demand for these foreign irons increased the price nearly 200 per cent. What was even worse, the dependence of England upon this foreign supply might possibly, at any moment, become a source of fatal weakness in war.

No doubt it was the perception of this critical situation which stimulated Henry Cort, both as a patriot and as an intelligent inventor, to risk his entire fortune and future upon the attempt to render his country independent of the world in a particular so essential to her prosperity. And the result of his enterprise may fairly be said to have altered profoundly the history of the world. Before 1785, England paid annually to Sweden alone £1,500,000 for wrought-iron. As the *London Times*[3] has said:

> "Then came the war, commercial embarrassment, depreciated paper, foreign prohibitions, and an overpowering and increasing demand for more and more iron. The inventions of Henry Cort carried us easily through this period of sharp trial, and, as his descendants allege, were the principal cause of our success."

A single instance may show how great was the revolution effected by Cort's improvements. Richard Crawshay, one of the first of Cort's licensees, who was making under the hammer barely 10 tons of bar iron per week, increased his product by the process (concerning which he wrote, "I took it from a Mr. Cort, who had a little mill at Fontley, in Hampshire") to 200

[3] July 29, 1856.

tons per week. But more significant still is the fact that, after the introduction of Cort's inventions, the Admiralty specifications, instead of excluding English iron, called for it.

Lord Sheffield said in 1786, only two years after the date of Cort's second patent:

"If Mr. Cort's very ingenious and meritorious improvements in the art of making and working iron, the steam-engine of Boulton and Watt, and Lord Dundonald's discovery of making coke, should all succeed, it is not asserting too much to say that the result will be more advantageous to Great Britain than the possession of the thirteen colonies of America."

And in 1865, nearly eighty years later, William Fairbairn said:

"Henry Cort's inventions have conferred an amount of wealth on Great Britain equal almost to six hundred millions sterling, and have given employment to six hundred thousand men."

The prophetic declaration of Lord Sheffield, quoted above, suggests another element in the military and commercial triumph of England which demands recognition—namely, the steam-engine of James Watt. The wonderful results of this improvement upon the Newcomen engine, first applied to hoist water from mines, have been abundantly, yet not too abundantly, celebrated. At various places in England, there are statues and memorials—half a dozen, at least—in honor of Watt; and another is projected, to be erected by British and American engineers, under the lead of Mr. Carnegie. They are not too many; he deserves them all. But it should not be forgotten that his friend Henry Cort furnished the most immediately—and, perhaps, we might even dare to say, the most permanently—important use for the steam-engine.[4]

[4] Mr. Charles H. Loring, in his Presidential Address before the American Society of Mechanical Engineers (1892), paid an eloquent tribute to the steam-engine, declaring it to be the great underlying cause of the wonderful progress of the last 100 years, and adding, "It is what no other machine ever was, the creator of physical power. The immortal inventor died without dreaming that he had placed on earth an infant Hercules, whose club, with an ever-increasing might, would batter down the institutions of preceding ages."

I cannot do better than repeat here, from my address on the history of the rolling-mill, delivered eight years later, and already cited in this paper, my comment upon this passage in Mr. Loring's address:

"I endorse that tribute, with this distinction only:—Watt's engine is the Hercules; but the rolling-mill is his club! Disarm him; take his club away—and

For what other application of that great motor can be deemed, even to-day, more important than the manufacture of the material of which itself consists, and through which, in countless forms and ways, its benefits are conferred upon men?

It would be ungrateful, as well as unjust, in setting forth the merits of Cort, to diminish in the least the credit and fame of Watt; for the appreciation and sympathy of Watt were of value untold to Cort, in his misfortunes, though these were of a sort which Watt himself had never been called to encounter. We are now considering, however, simply the value of Cort's two great inventions, not his personal experience; and on this point what I have already said may suffice, since it has been amply corroborated by the highest authorities.[5]

III. Cort's Reward.

This chapter also I shall make as brief as possible. Not long ago, I printed, for private circulation only, a "Review of the Case of Henry Cort," which I shall be glad to furnish to those who are seriously and sufficiently interested in its details. Here I shall confine myself to a bare statement of the essential facts.

1. For the purpose of obtaining additional capital, Cort

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how little, with his vast strength, can he do! The steam-engine may be 'what no other machine ever was, the creator of physical power;' but the rolling-mill has ever been the creator of the harness for using that power."

Discarding poetic expression, we must of course confess that the steam-engine does not create, but only liberates and transforms, the power already existing, latent, in the fuel it consumes; while the rolling-mill further transforms the same power into the molecular movement and consequent change of form of the metal it treats. But such comparisons serve no useful purpose. The overwhelming, providential fact is, that just at the time when Watt's steam-engine achieved recognition, Cort gave it the means of its most stupendous achievement, in driving a rolling-mill, and thereby inaugurating the most profound industrial revolution recorded in human history. Not steam alone, but steam and iron, have wrought that revolution.

[5] In addition to Percy and Webster, already cited, I may mention here the Duke of Devonshire, in his inaugural address at the first meeting of the Iron and Steel Institute in 1869 (*The Iron and Steel Institute Transactions*, vol. i., p. 5); Sir James Kitson (*Journal of the Iron and Steel Institute*, 1890, p. 392); Sir Lowthian Bell (*Ibid.*, p. 422); Sir William Roberts-Austen (*Ib.*, 1899, p. 21), and Mr. Andrew Carnegie (*Iron Age*, October 27, 1904, p. 45),—five Presidents of the Iron and Steel Institute (and Dr. Percy makes six), the body best qualified to give judgment in such a case. In fact, there is no difference of opinion on the subject among competent experts.

made a business agreement with one Adam Jellicoe, a Deputy Paymaster in the Navy, whose son Samuel he admitted to a partnership in his business, in consideration of which the father advanced to the firm considerable sums of money, taking as collateral security for Cort's share in the firm's debt, an assignment of Cort's patents. The arrangement was fair enough on its face; but the moneys advanced under it by Adam Jellicoe had been stolen from the government funds, with the collusion of the Treasurer, Mr. Dundas, afterwards Lord Melville. That Cort had any knowledge of this circumstance, there has never been either proof, accusation or suspicion.

2. The embezzlements of Adam Jellicoe had been carried on for years with the knowledge of the Treasurer and his Paymaster, one Trotter; but in 1788, under the pretense of a claim for repayment, these two forced him to surrender to them the Cort patents, which he held as collateral. Soon after (in 1789), Adam Jellicoe died; Lord Melville and Trotter instantly "discovered" his defalcation; and the latter, by a false affidavit, obtained an "extent" or summary attachment upon Cort's whole business and property.

3. At that time, the amount of the alleged defalcation of Jellicoe, for which Cort was thus arbitrarily made responsible, was only £27,500. Cort was then executing a Navy contract for £15,000; and was receiving in royalties £15,000 per annum. Yet Trotter swore that he verily believed the said Henry Cort to be "much decayed in his credit and in very embarrassed circumstances," so that His Majesty could not recover "the aforesaid debt" without more speedy means than the ordinary process of the court. Accordingly, Cort's property was seized.

4. After this seizure, the patents were never seen; the contracts with licensees likewise disappeared; and so far as can be discovered, no royalties were ever collected under them. These royalties alone would have repaid the whole of the alleged defalcation within two years. But nothing was done to administer the property; and Cort was thus at one blow deprived without redress of all his earthly possessions—forge, mill, patents and royalties—in a word, completely ruined.

The amount of the royalties already covered by agreements would have amounted, at the date of Cort's death, to £200,000. If Lord Melville, who confessed the possession of the agree-

ments, collected any part of this sum, he never accounted for it to the government.

5. Cort patiently sought to obtain justice, apparently without understanding the influences at work against him. He petitioned Trotter; he petitioned the Commissioners of the Navy; he petitioned Parliament; but was treated with indifference or insult. In 1794, a petition was addressed to the Prime Minister, William Pitt, praying that Cort might be appointed to some position in one of His Majesty's dockyards, or otherwise mercifully treated. This appeal brought a pension, netting him about £150 a year. After his death, his widow received net £100 a year, and later, on her decease, the two unmarried daughters received per annum £25 6s. each, "subject to reductions," or probably about £20. And this was doled out to a man from whom the government had taken more than £200,000, to repay a defalcation of £27,500, of which he was never even accused of having any knowledge!

6. But the greatest outrage was yet to come. In 1800, Henry Cort died; and as soon as he was out of the way, and his infant children were incapable of asserting their rights, Lord Melville presented a memorial to the Lords of the Treasury, setting forth the great merit and value of Cort's inventions, and asking on that account that *he himself* should be released from £25,000 of the Jellicoe defalcation, for which he held himself responsible! This petition was promptly granted; and presumably the Cort patents were turned over to the government and cancelled.

7. In 1803, Parliament appointed a Commission of Inquiry to examine the irregularities of the Treasurer, Lord Melville, and his Paymaster, Alexander Trotter; but these two worthies burned their accounts before they could be examined. The documents in the Cort case were destroyed with the rest. Upon his impeachment and trial (the amount of the defalcation of £25,000 having been made good in 1800, as stated above), Lord Melville was acquitted by a vote of 88 to 47 peers! In other words, a high official of the British Government had stolen money from its Treasury; the Government connived in his attempt to shield himself by robbing an innocent citizen, under the forms of law, abominably misused for that vile purpose; not only the patent-rights which the Government had

solemnly covenanted to protect, but also all other property of the innocent victim, were thus stolen from him by arbitrary process and without legal redress; and finally, as a climax of this performance, the original thief was rewarded, in consideration of the great value of the plunder, of which the Government of Great Britain then became the receiver!

8. It seems justly clear, however, that the Government treasury received no pecuniary benefit from this crime. It had been robbed by Lord Melville; and it credited him with the value of the proceeds of another robbery, which he transferred to it; but, so far as can be discovered, it did not realize any part of this value by collecting from Cort's licensees the royalties they were pledged to pay. Consequently, the beneficiaries of the outrage, next to the titled criminal (and to a far greater pecuniary extent), were those who were thereby released from the payment to Cort of the royalties (amounting to $1,000,000) which they had agreed to pay him, and, next to these, the manufacturers who, during the term of Cort's patents, used his inventions without even agreeing to pay royalty, and were never called upon for such payment. In view of the rapid extension of the practice of these inventions, thus prematurely thrown open to the public in total disregard of the inventor's guaranteed rights, it is a safe estimate that $2,000,000 more was saved to somebody. Whatever may have been the injustice of the summary proceeding by which Cort was robbed and ruined, it was clearly the duty of the Government, as the holder of the patent-rights and other property which it had seized, to administer them, and account for the resulting revenue. There is not the least trace that this was ever attempted; and in view of the revealed character of Lord Melville (which the whitewash applied by the Peers cannot hide), it is hard to believe that immunity from those payments of royalty under Cort's patents, which he could have enforced, was not bought from him by bribery.

Unfortunately, there is evidence, confirming this suspicion, of shameful treachery and falsehood on the part of some of the ironmasters who apparently profited by the remission of royalties and the practical cancellation of the Cort patents. In 1812 a petition for a parliamentary grant to Cort's family, in reward for his services to his country, was presented in the House of

Commons, with a recommendation from the Prince Regent. It was referred to a committee, of which the son of Lord Melville was a member. Two ironmasters were heard against the petition. Their evidence could have been easily overthrown; but the committee, excluding all evidence in contradiction or explanation of their statements, suddenly closed the investigation, and reported to the House that, though a considerable share of merit was due to Henry Cort, they could not satisfy themselves that it was sufficient to entitle the petitioner to a Parliamentary reward.

The testimony of the two false witnesses aroused much public indignation; and within a few weeks after the inquiry the chairman of the committee received a mass of evidence, concerning which he calmly said that "if it had been in time, the report of the committee would have been very different." Yet the children of Henry Cort were never able to get this evidence further considered. Even the recommendation of the committee of a grant to cover the expense of their futile petition for justice was ignored; and this amount was calmly charged against their penury.

9. The last chapter in this long story of wrong is the petition of Richard, the only surviving son of Henry Cort, presented to Parliament in 1856. Direct result it had none, so far as I know; indirectly, it was probably the cause of the grant to Richard Cort by Lord Palmerston, then premier, of an annuity of £50 (all previous Cort annuities, with their "reductions," having lapsed).

In connection with this revival of the question, several interesting publications appeared.[6]

IV. Our Unpaid Debt to Cort.

Samuel Smiles, in his biography of Cort, says:

"While the great ironmasters, by freely availing themselves of his inventions, have been adding estate to estate, the only estate secured by Henry Cort was the little domain of six feet by two in which he lies interred, in Hampstead churchyard."

[6] Among them, the article in the *London Times* of July 29, 1856, already cited, and the *Statement of the Claims of the Surviving Members of the Family of the late Henry Cort for National Compensation*, issued by the "Cort Committee," of which James Booth, LL.D., F.R.S., was chairman.

During a recent visit to England, I tried to find that "little domain." The sexton of the church knew nothing about it; but after the entry of the burial had been found by examination of the records, a renewed search discovered at last the grave. The headstone was weather-worn, and the inscription upon it hardly legible. It seemed as if inanimate things had conspired to perpetuate the obscurity and neglect wrought by the malice of man.

I have had this stone cleaned, and have obtained permission from the parish authorities to place in the Hampstead church a bronze tablet in honor of Henry Cort, which is represented in the frontispiece of this paper, from a photograph of the plaster-cast. The head is of life-size.

When our English brethren are ready to erect the larger monument which Henry Cort deserves, no doubt we shall all be glad to co-operate with them. Meanwhile, one American, at least, has had the opportunity and privilege of expressing in an enduring form, for the encouragement of other meritorious, though defeated heroes, the assurance that, sooner or later, history will do them justice, and succeeding generations will not let them be forgotten.

SECRETARY'S NOTE.—The tablet in honor of Henry Cort, to which Mr. Morgan here refers, was exhibited at the Engineers' Club, N. Y. City, before being shipped to the Hampstead church, for the wall of which it was destined.—R. W. R.

The Coal-Fields of Missouri.

BY B. F. BUSH, ST. LOUIS, MO.

(Lake Superior Meeting, September, 1904.)*

Area and Distribution.

The coal-fields of Missouri, situated in the northern and western portion of the State, are distributed, in whole or in part, over 57 counties, embracing an area estimated by Mr. Broadhead[1] to be practically 23,000 sq. miles, including small detached beds and outliers in St. Louis, St. Charles, Lincoln and Warren counties in the eastern part of the State. This area includes about 8,000 sq. miles of Upper or Barren Measures, 2,000 sq. miles of exposed Middle, and 12,420 sq. miles of exposed Lower Measures.

In a general way, the Missouri coal-field is bounded on the east by an imaginary line entering the northern part of the State just east of the 92d Meridian and passing across the State in a southwesterly direction to a point on the Kansas-Missouri line a little below the southern boundary of Barton county. An important part of the field overlaps this line and extends down the northern side of the Missouri river, underlying the greater part of Randolph, Howard, Boone and Audrain counties. In all, 34 counties produced coal during the year 1903. Fig. 1 shows the position of the coal-fields in Missouri.

Geological Features.

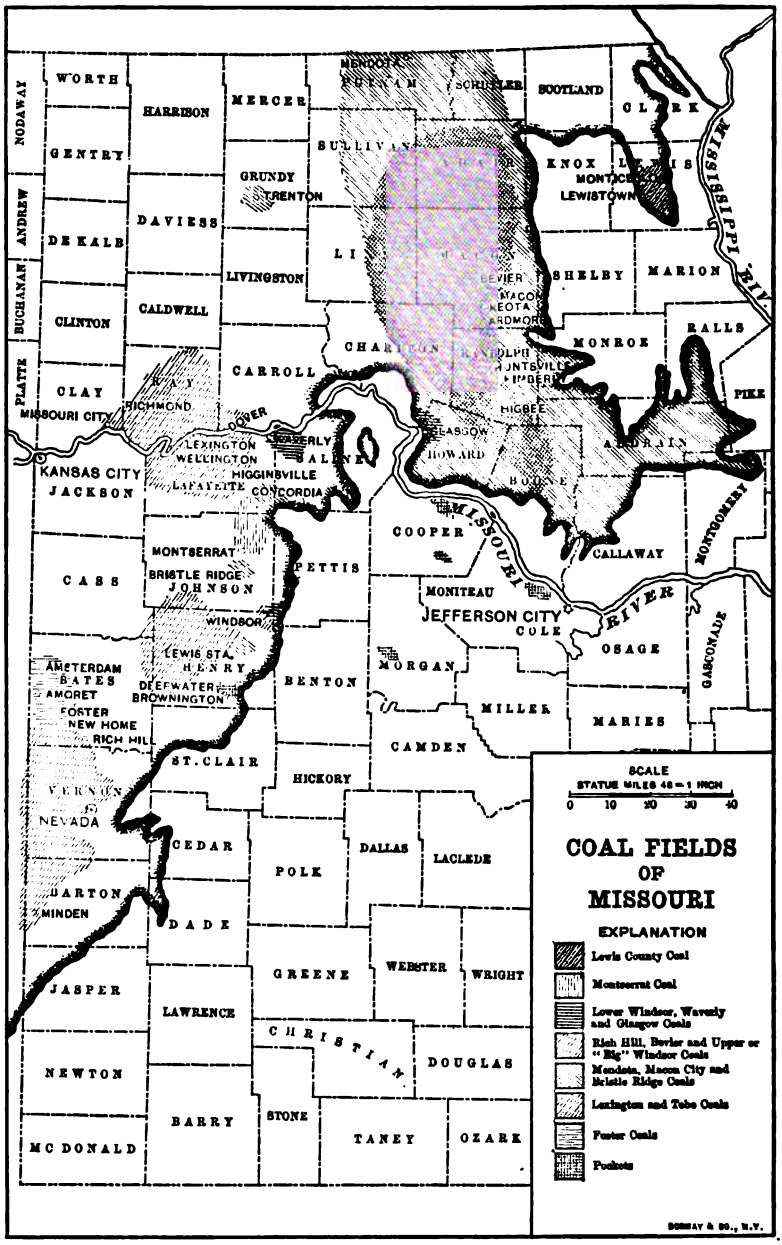
The Coal Measure formations, all of the Carboniferous age, rest unconformably upon the rocks of the sub-Carboniferous or Mississippian series, and, in some cases, upon the rocks of the Devonian and Silurian ages.

From their outcrop on the eastern and southeastern borders of the field, the Measures dip west to northwest at the rate of 10 ft. per mile, reaching a maximum thickness estimated by

* Presented at the Supplementary Session of the Institute in St. Louis.

[1] *Report of the Missouri Geological Survey*, 1894, p. 355.

FIG. 1.



MAP OF THE COAL-FIELDS OF MISSOURI.

Mr. Arthur Winslow[2] at 1,900 ft. A section, embracing the successive formations from the basal sandstone to the argillaceous limestone cap-rock of the Arkoe, Maitland and Quitman coal-horizons in Nodaway county, consists of interbedded limestones, sandstones, shales, fire-clays and the coals themselves, which in their fullest development are 29 in number. The coal-seams vary in thickness from 1 in. to 5 ft., often thinning out and disappearing entirely. The limestones are absent in the lower formations, but are quite prominent in the Upper Measures. The shales, in their various phases of argillaceous, bituminous, calcareous and arenaceous, are most prominent. The sandstones, although prominent in the marginal areas, are not prominent in the central portion of the field. The coal-beds are generally thin and variable in thickness and are sometimes represented by only a carbonaceous streak. The beds are generally underlain by clay and usually covered by black shales, or by a gray or drab clay-shale. For various reasons, the coal-formations have been divided into three classes, the Upper, the Middle and the Lower Coal-Measures.

Lower Coal-Measures.—The Lower Coal-Measures embrace eight coal-horizons resting on the basal sandstone and capped by the Mahoning sandstone. There is no limestone in these Measures. Two only of the eight horizons are sufficiently developed to be workable. The first is productive in a small way in portions of Lewis and Clark counties. The second horizon has yielded a considerable quantity of coal at Montserrat. With these two exceptions, these Measures are unproductive in Missouri, although all the coal from Arkansas and Indian Territory and the Big Muddy coal of Illinois, occur in the Lower Measures.

Middle Coal-Measures.—The Middle Coal-Measures embrace all the formations between the Mahoning sandstone and the Bethany Falls limestone. They contain 12 well-defined coal-horizons from which almost the entire output of the State is obtained. All the horizons, with the exception of the second, are capped by limestone rocks, generally of an argillaceous nature. All of the first four horizons are productive. From the first, the Windsor, Waverly and Glasgow coals are mined. In the second, the Rich Hill, Bevier and Windsor coals occur.

[2] *Report of the Missouri Geological Survey*, 1891, p. 23.

The third yields the Mendota, the Macon City, the Brush Creek and the Bristle Ridge coals; from the fourth, the Lexington coal, so extensively mined throughout Ray, Lafayette and Clay counties, is obtained. The eight remaining horizons are unproductive, with the exception of the tenth, which is mined in one locality in a small way.

Upper Coal-Measures.—The Upper or Barren Coal-Measures follow the Bethany Falls limestone and include nine coal-horizons, all of which are very thin and of little economic value, with the exception of the last, which has been mined a little in Nodaway and Atchison counties. The fifth also has been mined at times, but to a very limited extent.

The beds are generally regular and are associated with numerous shales, generally of the argillaceous and calcareous type, and with thick and persistent limestones, the Nodaway, St. Joe, Kansas City, Weston and Parkville limestones all occurring in these Measures. The Measures are capped by the Quitman limestone.

Geographical Features.

The coal-fields of Missouri are composed of various small beds, nowhere of any great extent and, in many cases, either too thin or occurring at too great a depth to be worked at a profit. The greater part of the coal mined to date has been taken from the marginal area where the beds are most numerous and lie nearest to the surface. These have to a great extent been exhausted and in the near future it will be necessary to look to the thinner and deeper beds. Under the present conditions, they cannot be worked profitably, and with such an abundant and cheap supply at hand on all sides, it is doubtful whether they ever will be mined.

Montserrat Coal.—The Montserrat coal underlies the greater portion of Johnson county and a portion of Lafayette county, and, at one time, was extensively mined near the town of Montserrat. The inferior quality of the coal, however, caused its abandonment some years ago, and at the present time it is only mined on a small scale. In thickness it runs as high as 5 ft., the lower 3 ft., however, containing almost all of the good coal. It slacks quickly and contains a large proportion of pyrite.

Waverly Coal.—Waverly coal is found in the extreme northeastern portion of Lafayette county, near the town of Waverly. The extent of this field is limited and of a very doubtful nature, while the bed is extremely difficult to approach, on account of the physical features of the country. The Coal-Measures, as near as can be learned from developments, show a workable vein, more or less faulted, about 4 ft. thick, with only one or two small bands of slate, which can be removed in mining. The coal rests on rock and has a good roof. A little gas is present and the room-and-pillar system of mining is needed to extract it. The coal is of a poor bituminous variety, low in fixed carbon and high in ash. A large percentage of sulphur, in the form of pyrite, also is present.

A coal of the same general characteristics is mined at Windsor and at Glasgow, which is believed to be the same bed. It is found also in the Bevier district, where it is known as the Eureka bed. It is excessively high in ash and is not mined in that field.

Bevier Coal-Field.—Bevier coal has the widest distribution of all the Missouri coals and from this bed the greater part of the State's tonnage is derived. It has been extensively mined for years at Huntsville, Ardmore and Bevier. It spreads out over Macon, Linn, Chariton, Randolph and parts of the adjoining counties. The coal varies in thickness from 4 to 6 ft. In the northern portion of the field it lies at an average depth of from 50 to 70 ft. from the surface, but deepens towards the south. The bed is cut into by the Moberly sandstone and also by the larger drainage-channel, and in many places is very difficult to approach. In the vicinity of Bevier the coal is mined by shafts. In the southern districts, however, advantage is taken of its numerous outcrops along the hillsides and the mines are principally drifts and stopes. The coal is mined by the room-and-pillar system. The roof is generally of a shaly character, grading at times into a sandstone. The sandstone affords a good roof, but the shale is more or less treacherous and requires considerable propping. The thickness of the seam is frequently diminished by horse-backs and rolls, which sometimes entirely cut out the coal.

The coal is mined extensively at Bevier, Huntsville, Higbee, Ardmore, Kimberly and Keoka. There are also many small

mines that are worked intermittently. The coal itself is high in ash, moderate in moisture- and sulphur-content, and is the only coal that has successfully met the competition of the coals from neighboring States. The same coal is found at Rich Hill in Missouri, and at Leavenworth, Kansas.

Rich Hill Coal.—The Rich Hill coal-field lying in the southwestern portion of Missouri includes the western halves of Barton and Vernon counties, and the southern portion of Bates county. Near Rich Hill the coal has been found in numerous small basins or lenses of from 100 to 500 acres in area, closely adjoining each other. The coal varies in thickness from 2.5 to 5 ft. No coal has been found under Rich Hill, but there are two series of basins, one to the northwest and the other to the south of the town. The theory has been advanced that this coal was deposited in many small adjoining lakes.

The Rich Hill coal-field, although practically exhausted at the present time, was, for many years, the most productive one in the State and had few if any equals in the coal-fields of the West. The country around Rich Hill, for a radius of 15 miles, has been thoroughly prospected by the Rich Hill Coal Mining Co., but no coal was found to which it would be profitable to sink a shaft, with the sole exception of the New Home field, comprising a small basin of approximately 600 acres. This basin would be expensive to operate, owing to the irregularities in the undulating course of the coal-strata.

The principal operations in southern Missouri are at Minden, where the vein, which crops out just east of Minden, is the same as that so extensively mined throughout Crawford and Cherokee counties, Kan.

The bed averages about 42 in. thick and has a good roof of block-slate and a fire-clay bottom. The coal-seam has a uniform dip, frequently cut by clay-slips and horse-backs. Its quality ranks first in Missouri.

The Foster coal-bed is the only other one mined in this field. It averages about 30 in. thick, is very faulty and irregular and has a very treacherous roof. It is mined in small quantities near Worland, Amsterdam, Amoret and Foster. On account of the shallowness of the covering, most of the mining is done by stripping. The coal is low-grade, soft, fragile and slacks quickly on exposure to the atmosphere.

The geological position of this bed has not been fully determined.

Mendota Coal-Field.—Mendota coal is the same vein that is extensively mined in southern Iowa. It underlies the eastern portion of Putnam county and the western portion of Adair and Schuyler counties, Mo., but its southern limit is not well defined. The bed, varying in thickness from 2 to 3.5 ft., is usually separated into three layers by pronounced clay- or shale-partings. The roof is generally of shale or slate overlain by 4 ft. of limestone. In many places, however, a soapstone has been deposited between the shale or slate, and the coal generally giving a roof that is bad. The bottom is a hard fire-clay. The region is hilly and the coal is frequently mined by drifting into the outcrops along the hill-sides. In other cases it is mined by the room-and-pillar system. The coal-bed is considerably cut up by faults and rolls. The coal lies at an average depth of 70 ft. below the higher country-rock.

A bed, thought to be the same as the Mendota bed, is mined at Mexico in Audrain county, Macon City in Macon county, Brush Creek in Jackson county and near Concordia in Lafayette county. It is, however, too thin to be mined to any great extent. At Macon City the coal is of a very good quality and may yield a fairly good supply in the future.

Lexington Coal-Bed.—The Lexington coal-bed underlies the greater portion of Ray and Lafayette counties and a portion of Clay county. It lies from 200 to 300 ft. above the Waverly coal, and is found above the water-level at all points east of Lexington along the Missouri river. West of Wellington, it is found below the water-level. This vein, probably the first worked west of the Mississippi river, has been worked in the vicinity of Lexington for more than 75 years, and at one time a considerable quantity of coal was shipped from there by steamboats to places above and below on the Missouri river.

The opinion has been generally accepted that the entire Lafayette county was underlain by coal, but this is not true. The coal is entirely cut out by the Warrensburg sandstone, east of Lexington. This belt of sandstone reaches a maximum width of about 8 miles near the Missouri river, and tapers as it extends back towards Higginsville. Like a huge octopus, its many arms extend in every direction and cut out the coal on all sides.

The bed is cut also by valleys of the large water-courses, and in many places the coal is entirely absent.

All the available territory fronting on the river around Lexington has been practically exhausted, although a small tonnage is still being mined west of the town.

There are no undeveloped coal-fields available at present to transportation-facilities, and those more remote are not sufficiently large to justify the expense of building railroads to reach them. At the present rate of consumption, profitable operations in this coal-field will be brought to a close in perhaps 15 years, although isolated openings may be subsequently operated.

The new mines of the Lexington Coal and Mining Co., south of Lexington, tap the largest and best field in the territory, which is continued for 5 or 6 miles southward of the mines and has an average width of from 3 to 4 miles. Several additional new mines could be operated in this field, all of which would prove profitable, but beyond that there is no field where mining-operations can be pursued on a large scale.

The coal-seam varying in thickness from 16 to 24 in. has a clay-bottom, and is overlain by a block-slate which comes down with the coal. Above the slate is a solid blue limestone roof. The conditions are almost perfect for the long-wall system of mining.

The coal is of good quality and is well adapted to domestic and steam-purposes. A portion of the coal mined southwest of Higginsville, known as "farmers' coal," sells at a slightly higher rate than the regular Lexington coal. A peculiar feature of this coal is its saturation or impregnation with a thick, black, oily substance, resembling tar. The coal is very rich and pure and finds a ready sale for domestic use. For steam-purposes, however, it is no better than Lexington coal, and as the vein here is only 17 in. thick, it is more expensive to mine.

To the east of the barren belt formed by the Warrensburg sandstone, coal is again found near Dover, 11 miles east of Lexington. The field, originally small, has been further reduced by erosion. The coal is the Lexington seam with the same general characteristics, varying in some details, but not in a manner to lessen the cost of production. The quantity of

the coal is good and equal to the best Lexington coal. The seam varies in thickness from 12 to 28 or 30 in., and is subject to rolls. The roof, though not so satisfactory as at Lexington, is not troublesome. The bottom is soft and the digging easy, the undercutting being done in a clay underlying the coal-seam. The cost of props offsets the easier mining and the increased thickness. This same coal is found in Ray county at the same relative level, and is operated on the same general plan. This may also be said of the coal in Clay county. The coal-seam dips very rapidly as the western boundaries of Ray and Lafayette counties are approached, and at Randolph Bluffs, 4 miles northeast of Kansas City, it lies at about 400 ft. below the surface.

Jordon Coal-Field.—The Jordan coal-bed is of very limited extent and is mined principally in and about the territory of Deepwater in Henry county. The coal is intercalated with beds of slate, and both roof and bottom are of shale. In many places the roof is poor, the water is troublesome, and the quality of coal not high. The vein, which is stated in the mine inspector's report to be 3 ft. thick, will be found considerably less when mined. The coal lies at an average depth of about 35 ft. below the surface.

Tebo Coal-Field.—Tebo coal is found chiefly in Henry county, with a small area overlapping into Bates and Johnson counties. This coal is also intercalated with slate and bands which have to be separated from it before the coal is ready for market. The vein has a good roof and varies in thickness from 20 to 30 in. The relation of this bed to the Lexington bed has not been fully determined.

Various Coal-Pockets.—Numerous irregular isolated deposits of coal are found in Missouri, usually just beyond the marginal area of the regular coal-beds. These irregular deposits are often of great thickness, as compared with the regular Coal Measures, having been found as thick as 75 feet. They are found at various depths, and, in many ways, they appear to be remnants of some former field that has been removed by erosion.

Geologists differ in their opinions regarding the origin of this coal. In many places it seems to be of a later formation than the regular beds. The coal is generally of poor quality,

approaching lignite in character, and is often of the cannel type. Occasionally it also carries impurities in the form of pyrite, lead and zinc sulphides, and other kindred minerals. A curious occurrence of zinc blende is found with the coal in numerous places in Cooper, Morgan, Cole and Saline counties. I have heard of only one locality where a similar phenomenon has been observed. The zinc blende fills the larger vertical planes of the coal, and has evidently been deposited from solution.

These irregular coal-beds usually occur in the banks of streams and ravines and exhibit signs of disturbances. Although unconnected with one another, they bear the same general characteristics. They have never been mined to any great extent, but they have furnished a considerable quantity of coal for local use.

In connection with the theory that these irregular deposits are remnants of other fields, an unusual occurrence was noted at Lexington some years ago. Near the Lexington depot a shaft was sunk at a point where it was 100 ft. deep to the regular Lexington bed. Above the coal was a layer or two of thick limestone-rock. Above the rock were found drift, logs, lumps of coal weighing as much as from 300 to 400 lb., and scattered boulders. The whole rested in a bed of quicksand, showing that there had evidently been a disturbance, in which the regular Lexington coal had been torn up at some point and evidently re-deposited by the agency of water.

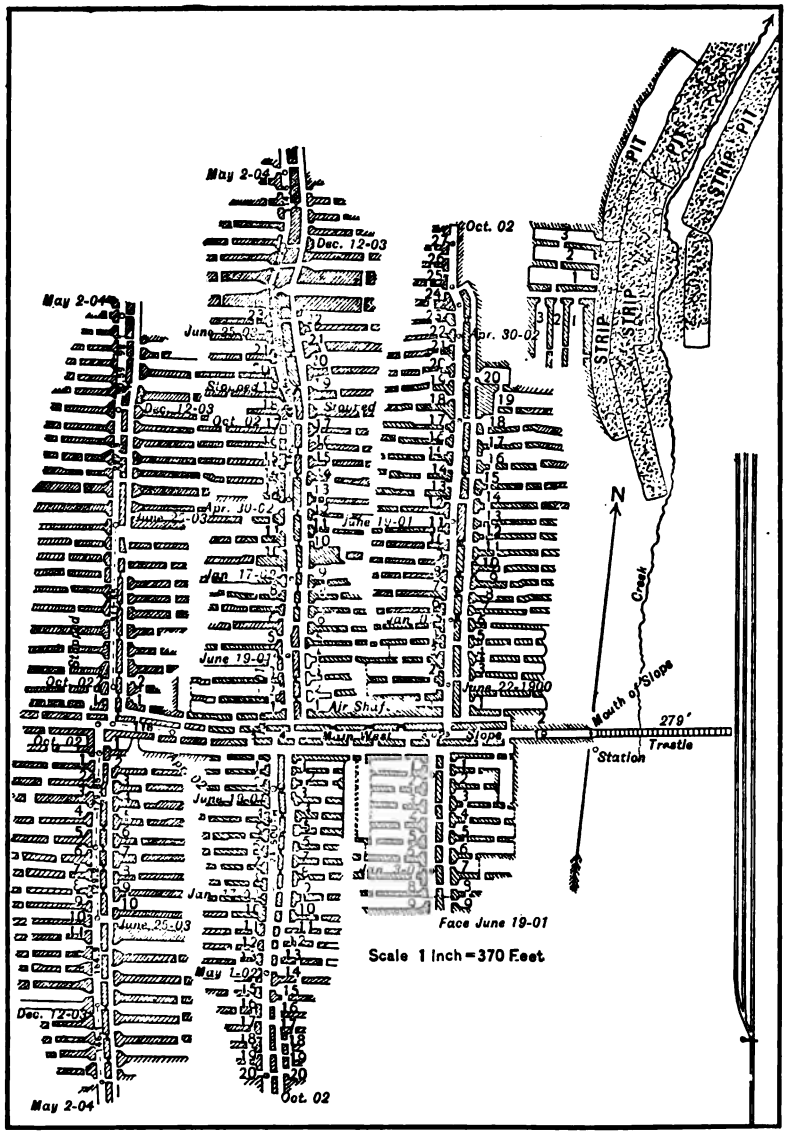
Methods of Mining the Coal.

Stripping.—Along the outcrop of the various beds, and in many other places where the physical conditions of the country permit, a considerable tonnage is produced from strippings. In several of the beds, from want of satisfactory roof-conditions, this is the only method by which the coal can be properly won.

In general, most of the work is carried on in the fall and winter, when the market-conditions are most favorable, and teams and labor are in least demand for other purposes. The coal is of inferior value and slacks quickly on exposure and handling. There were 47 pits in operation during 1903, in each of which more than 10 men were employed, and, in addition, many small pits were worked intermittently.

Room-and-Pillar.—This type of mining is the most common in Missouri, there being 164 mines so worked in 1903. In

FIG. 2.



MAP OF ROOM-AND-PILLAR WORKING AND OPEN-PIT MINING AT MINDEN, BARTON COUNTY.

Missouri the shaft-pillars vary from 100 to 400 ft. square, depending on the nature of the roof, the overlaying strata and the

depth at which the coal is mined. The entry-pillars are generally about 20 ft. wide and rooms are turned off at intervals of about 30 ft., leaving 10-ft. pillars between rooms. Cross-entries are turned about 300 ft. apart. The double-entry system is required by law with cross-cuts every 50 ft. Fig. 2 shows the method of room-and-pillar working and open-pit mining at Minden, Barton county.

The coal is generally shot off the solid, although in many places, especially in the Mendota field, the coal is undermined, cut on the sides and wedged down with a pick, no powder being used.

Long-wall.—The long-wall system, which offers the fullest advantages in mining the thinner seams of Missouri, is generally used in the thin seams. The Lexington coal-bed is extensively mined by this method, both in Lafayette and Ray counties, where the general thickness of the vein ranges from 16 to 24 in. By this system all of the coal is extracted in the best condition. Ventilation is comparatively simple, and the haulage-distance for a given output is relatively short. Few roads are required to be kept open and the coal is easily worked. The cost of timber, track and other equipment is minimized and that of powder entirely saved.

The long-wall method of mining followed by the Lexington Coal Mining Co. at the Valley mine is shown in Fig. 3.

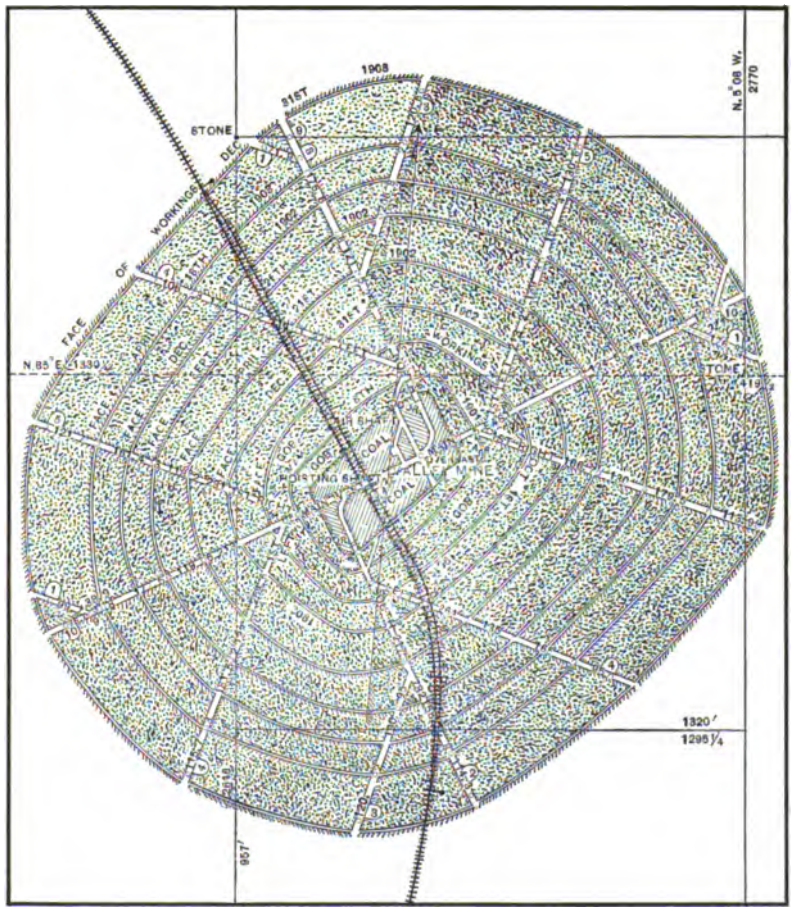
In order to understand properly the long-wall method of mining at the shaft of the Valley mine, Lexington, Mo., it is necessary to know something of the strata immediately above and below the coal. A section at the bottom of the shaft shows:

	Inches.
Limestone,	84
Roof-slate,	2
Block-slate,	6
Band-slate,	3
Coal,	20
Mining-dirt,	1
Sulphur-rock,	3
Fire-clay,	19
Limestone-bottom,	...

In opening up the Valley mine, two main entries were driven east and west from the shaft. At a distance of 150 ft. from the shaft, the main entry north was turned off the main east and

the main entry south off the main west. At distances 75, and 125 ft. further, the branch-entries were also turned at an angle of 45 degrees. Cross-cuts were then driven between these entries, leaving a pillar to support the bottom, 200 ft. by 450 ft., in the shape of a parallelogram, the ends running north and

FIG. 3.



MAP OF THE LONG-WALL WORKINGS AT THE VALLEY MINE, LEXINGTON, MO.

south, but the sides conforming to the bulk of the pillar around the main north and main south entries.

An air-course connects the air-shaft with the main north and main south entries, passing across the main east entry by means of an overcast which is tunneled above the limestone, or "head" rock as it is locally termed. The exterior ribs of the

cross-cut form a long wall-face, which is worked in the following manner:

The fire-clay below the coal is removed by electric coal-cutters to a depth of from 2.5 to 3 ft., the cut being about 6 in. wide. The coal is spragged and held until the machine has passed 30 or 40 ft. The sprags are then removed and the gradual increase of roof-pressure upon the face breaks down the coal in large lumps. The two slates above and the sulphur-rock below come down with the coal, but separate from it in falling. Four machines are operated at the present time, making a continuous circuit around the face. They are operated night and day, and travel fast enough to mine each man's coal every other day. The miners spend the interval of time in loading the coal, packing and gobbing the walls with rock and slate, and in cleaning up their places.

The movable face-track resting on iron ties is pushed forward by each man, and an extra or fractional part of a rail-length inserted and the miner is again ready for the machine.

In this manner the face gradually advances and the roof settles behind it on the packing in the gob. The entries advance with the face and are kept open by gobbed walls and packings. When the main entries have advanced about 650 ft. from the last branch entry, a triangular gob-wall is built and another parallel cross-entry is turned. This arrangement confines the distance between entries at the face to about 450 feet.

On haulage-roads it is necessary to "brush" the roof to within 50 ft. of the face. The roof-slate and a portion of the overhead limestone are shot down and the fire-clay removed, giving an entry of about 6 ft. 8 in. Each miner is given a working-place of from 30 to 75 ft., according to ability. The distance between the packing and the face is about 7 ft. The hauling is done by mules. The mine is ventilated by a 12-ft. fan; 150 men are employed, producing 600 tons of coal per day, which is equivalent to 4 tons per man.

Statistics of Production.

The annual production of coal in Missouri during the past 20 years has fluctuated between 2,000,000 and 4,000,000 tons. The industry showed a steady growth up to 1888, reaching a tonnage that year of 3,909,967 tons. The following year the

tonnage decreased more than 1,500,000 tons. In the period of five years following, a steady increase was maintained, and in 1893 the output was again increased to 3,000,000 tons. In 1894, however, it showed a decrease of 800,000 tons. Since 1894, the increase has been comparatively regular, and in 1902, the 4,000,000 mark was passed. During this period, the average price per ton at the mines has varied from $1.10 to $1.58 per ton, the former price being paid in 1897 and 1898, the latter price during 1903.

The lowest annual tonnage recorded for 20 years was in 1886, when the output amounted to only 1,800,000 tons. The highest production was reached in 1903, the record being 4,265,328 tons. In this year Missouri ranked fourteenth among the coal-producing States.

There were 348 corporations and individual companies mining coal in Missouri during 1903, of which one company produced one-fourth of the total output and ten companies considerably more than one-half. Eighty per cent. of the total quantity of coal mined in Missouri during 1903 was produced by nine counties as follows: Macon, 1,198,133; Lafayette, 539,612; Randolph, 450,181; Bates, 359,061; Adair, 312,403; Ray, 280,162; Vernon, 207,125; Barton, 200,433; Putnam, 125,543 tons. No other countries produced as much as 100,000 tons.

Character of Missouri Coals.

All of the coals of Missouri, with the exception of a few local deposits of cannel-coal, are of a distinctly bituminous nature, and are not of high grade. They are not adapted to the manufacture of gas on account of the large proportion of pyrite and ash present. They are comparatively soft and suffer quickly from exposure or excessive handling. They are essentially steam-coals and are mainly used for locomotives and household-purposes.

Blast-Furnace Plant of the "Elba" Societa Anonima di Miniere e di Alti Fórni at Portoferraio, Elba.

BY CARLO MASSA, GENOA, ITALY.

(Lake Superior Meeting, September, 1904.)

THE Società Anonima di Miniere e di Alti Fórni, familiarly known as the "Elba Company," works the rich ore-deposits of the Island of Elba under a long-term contract with the Italian Government. The ore-deposits are scattered over the eastern and southeastern coast; but as these parts of the island are not well provided with ports, the blast-furnaces were built at Portoferraio, the principal town of the island, situated on the northern shore, and possessing a magnificent natural harbor where vessels even as large as the battleships of the Italian navy can comfortably anchor. In the selection of Portoferraio the cost of transporting the iron-ore from the different mines to the blast-furnaces has been reduced to a minimum, and shelter has been provided for the company's flotilla of tugs and barges, as well as for the steamers carrying coal and pig-iron.

The whole plant, of which a general view is given in Fig. 1, comprises:

1. *A discharging-pier* with unloading-cranes. The ore and coal are transported from this pier by means of an aërial cable-railway to the coal- and ore-storage yards. This cable-railway is supported by an iron bridge extending from the pier to the shore and by a wooden trestle extending from the shore to the yards.

2. *Blast-furnaces*, two in number, with all the accessory apparatus, such as blowing-engines, dry purifiers, Cowper hot-blast stoves, hoists, etc.

3. *Coke-ovens*, two batteries, with coal-bin, crushers and boilers.

4. *Engine-house* containing the blast-engines and the electric generating-station.

5. *Pumping-plant.*

6. *Narrow-gauge railways.*

The areas covered by some of the sections of the works are:— Discharging-pier and trestles, 90,000; Blast-furnaces, purifiers, hot-blast stoves, 31,000; coke-ovens, 42,000; and engine-house, 19,250 sq. feet.

Electricity is extensively used throughout the works. The entire plant is lighted exclusively by electricity, there being in service about 300 incandescent lamps (of 10 and 16 candle-power) operating at 250 volts on the two-wire system, and 40 arc lamps operating at 42 volts with a current of 10 amperes. In addition to lighting, electricity is the only motive power used except for the blowing-engines; the hoists, cranes, centrifugal pumps, Theisen gas-purifiers and furnace-bell being worked electrically.

A short description of the different parts of the works and some numerical data are given below:

I. Discharging-Pier and Aërial Cable-Railway.

The entire plant, serving to unload the arriving ships and to convey the iron-ore and coal to the various parts of the works, consists of three different sections:

1. *Discharging-pier*, properly called, which is a horizontal iron structure, 340 ft. long and 53 ft. wide, with wooden platform placed at a height of about 10 ft. above the sea-level. This structure is supported by piles formed of stone and concrete, varying in depth from 43 to 56 ft. below sea-level. On this pier is placed the entire machinery necessary for working the aërial cable-railway and 8 hoisting-cranes. The latter are arranged in two groups of four placed on each side of the pier, so that the unloading of ships may proceed simultaneously on both sides. One side is reserved for the coal-steamers, the other for the iron-ore barges, and each crane can unload 25 tons per hour. The coal or iron-ore unloaded from the ships is dumped into hoppers from which the buckets of the cable-railway are filled. These buckets have a capacity of 2,200 lb. on the iron-ore side of the pier and of 1,650 lb. on the coal side. They are filled and attached automatically to the traveling-cable and run at a speed of about 4 km. (2.5 miles) per hour. The necessary motive power is furnished by three 50-h.p. Westinghouse four-pole compound-wound motors running at 900 rev. per min. and at a tension of 220 volts. Each of the two hoisting-crane

groups is worked by one motor by means of a belt-pulley and countershaft; while the third motor runs the aërial cable-railway by means of simple belt-transmission.

2. *Connecting-bridge*, which is a lattice-iron structure having a slope of about 0.07 per cent., that serves to support the cable-railway and also compensates the difference between the level of the pier and that of the trestles.

3. *Trestles*, which are at the end of the connecting-bridge, support the two aërial cable-railways that branch off in two different directions, one to the ore-storage yard the other to the coal-yard, traveling over wooden trestles having a height of 52 ft. The buckets (or cars) are tripped at the proper places and discharge their contents on to the heaps below.

II. Blast-Furnaces and Accessory Apparatus.

Originally the two blast-furnaces of the plant were built on the same pattern each of the following dimensions:—Total height, 74; diameter of throat, 15; diameter of bosh, 20, and diameter of crucible, 10 feet.

Each furnace had six tuyeres and was intended to produce 150 tons of pig-iron per day. The normal output of each furnace, however, was 200 tons in 24 hours.

Later, it was decided to modify the lines of furnace No. 1, so as to increase its capacity to 350 tons. The dimensions after remodeling were:—Diameter of throat, 15; of bosh, 21, and of crucible, 12 feet. The number of tuyeres was increased to 12, which were uniformly distributed around the furnace, and a Sahlin patent spiral-cooled bosh-jacket was added. The operation of this furnace has proved satisfactory and shortly furnace No. 2 will be put out of blast and remodeled in like manner.

The two blast-furnaces work with different kinds of ore. One uses the more phosphoric ores and produces high-grade foundry-iron; while the other uses very pure ores (containing from 0.007 to 0.010 per cent. of phosphorus), and produces first-class hematite-iron.

The bells, closing the hoppers at the furnace-tops, are worked electrically by small series-wound motors controlled by tramway-type controllers. The raw materials—coke, ore and lime-

FIG. 1.—GENERAL VIEW OF THE BLAST-FURNACE PLANT AT PORTOFERRAIO, ITALY.

FIG. 2.—VIEW OF COKE-OVENS AND FURNACES AT PORTOFERRAIO, ITALY.

stone—are hoisted electrically and distributed in the hopper by hand.

A Krupp electric crusher is used to crush the hardest ores before charging them into the furnace.

The accessory apparatus comprises :—

1. *Hoists* for raising the raw materials to the top of the furnaces.

2. *Dry purifiers*, in which the dust carried by the gases is deposited.

3. *Theisen gas-purifiers*, in which these gases are washed.

4. *Cowper hot-blast stoves*, in which the cold air-blast coming from the blast-engines is heated.

The building for the hoisting-machinery, placed between the two furnaces, contains two machines, each capable of lifting in 30 seconds two loaded wagons (weighing from 1,750 to 2,200 lb. each) to the top of the furnaces. Each hoist is composed of two drums rotating in opposite directions on which the cables supporting the hoisting-cages are wound. These drums are driven by Westinghouse four-pole series-wound railway-type enclosed motors of 50-h.p. capacity operated by commutator-type controllers. The movement of the motors is transmitted to the drums by means of double gearing.

The furnace-gases are collected at the top of the furnace by a perpendicular tube passing through the center of the bell, and thence led through a downcomer to the dry purifiers, of which there are four for each furnace. The greater part of ore and carbon-particles carried by the gases is deposited here; but the dry purification is still carried on inside the gas-main, which is provided with dust-catchers throughout its entire length. Before being admitted to the gas-engines, the gases undergo a wet purification in the Theisen gas-cleaning apparatus and afterwards in a series of water-scrubbers where the temperature of the gas is reduced to normal. In the works there are two Theisen gas-cleaning apparatus direct-connected to Westinghouse six-pole shunt-wound motors of 100-h.p. capacity running at a speed of 520 rev. per minute.

The cold air-blast furnished by the different blowing-engines is heated in the Cowper stoves to a temperature of about 1,300° F. and then driven to the furnace tuyeres. There are four Cowper stoves for each furnace.

III. Coke-Ovens.

There are two batteries, covering together an area of about 42,000 sq. ft., of coke-ovens, each containing 52 retort-ovens of the following dimensions:—Length, 33 ft.; average width, 1 ft. 7 in., and height, 4 ft. 11 inches.

Each oven can yield 7,000 lb. of coke in a 24-hr. run, and the daily capacity of both batteries is equivalent to about 325 tons of coke.

To the two batteries are annexed:

1. A coal-storage bin covering a surface of 38,800 sq. feet.
2. A mill for crushing and pulverizing the coal, driven by a 75-h.p. steam-engine.
3. Three Cornwall boilers of 1,075 sq. ft. heating-surface and two fires-tubes each, using as fuel the gases produced by a set of 20 coke-ovens. The gases of the remaining ovens, up to the present, have not been utilized.

A view of the coke-ovens and boilers is given in Fig. 2.

Work will be commenced shortly on an additional battery of 80 by-product coke-ovens having an aggregate capacity of 400 tons per day.

IV. Engine-House.

The engine-house, which is a stone building 82 ft. wide and 236 ft. long, may be divided into two distinct parts: The first containing all the blowing-engines; the second containing the electric generating-station; both parts being united under the same roof.

The blowing-machinery comprises two gas-engines and one steam-engine.

Each gas-engine is of 500-h.p. normal capacity and is of the following dimensions:—Diameter of cylinder, 67 in.; stroke, 55 in., and speed, 80 rev. per minute.

The maximum output of these engines is 17,500 cu. ft. of air per min. and the pressure of blast 8 lb. per sq. inch. These gas blowing-engines are of the Cockerill-Delamare type.

The steam blowing-engine, built by the Meuse Works of Liege, is a 1,200-h.p. horizontal compound-condensing engine of the following charactertstics:—Diameter of blowing-cylinder, 84 in.; of high-pressure steam-cylinder, 39 in.; of low-pressure steam-cylinder, 63 in.; the stroke is 59 in.; the initial pressure is 132 lb.; the maximum speed is 42 rev. per minute.

The air-output is 750 cu. ft. per revolution. At that rate the blast-pressure can be raised as high as 10 lb. To this engine is annexed a sea-water cooled surface-condenser with a cooling-area of 2,400 sq. ft. having the necessary pumps.

A fourth gas blowing-engine now being erected is a 1,500-h.p. Cockerill-Delamare double-effect gas blowing-engine, of the following proportions:—Diameter of gas-cylinder, 51 in.; of air-cylinder, 87 in.; the stroke is 55 in.; the speed is 80 rev. per min., and the air-output at 12-lb. pressure is 28,000 cu. feet.

The electric generating-station consists of three belt-driven continuous-current 100-kw. dynamos connected to gas-engines, and one belt-driven continuous-current kw. dynamo connected to one of the celebrated Tosi engines.

Each of the three gas-engines, of the Cockerill system, has a capacity of 200 h.p. and is of the following proportions:—Diameter of cylinder, 33 in.; stroke, 39 in., and speed, 105 rev. per minute.

The steam-engine, built by the Tosi firm, of Legnano, is a horizontal compound-condensing engine with tandem cylinders (Woolf engine); its capacity is 280 h.p. normal and 370 h.p. maximum. Its principal data are: Diameter of high-pressure cylinder, 18 in.; of low-pressure cylinder, 30 in.; the stroke is 41 in., and the speed, 80 rev. per minute. Each engine is provided with a sea-water cooled surface-condenser and the necessary pumps.

The four dynamos, of Westinghouse construction, are all six-pole compound generators. The speed of the 100-kw. units is 550 rev. per min., that of the 200-kw. unit, 450 rev. per minute. They generate current at a tension of 250 volts, which is the general voltage of the entire continuous-current distribution. There are two mains only.

The ignition-circuit of the gas-motors is fed from two small motor-generators reducing the voltage from 250 to 4 volts; the 250-volt bi-polar-motors of 0.5-h.p. capacity are direct connected to the 0.33-h.p. dynamos generating-current at 4 volts; the common speed is 1,800 rev. per min., and a small storage-battery is used as reserve to these sets. The continuous current thus generated is rendered discontinuous in the gas-engines by Ruhmkorff coils.

The engine-house still contains an electric transforming

sub-station where the low-tension continuous-current is converted into high-voltage alternating-current and transmitted over a line of two miles to a pumping-plant which supplies water to the boilers. This sub-station consists, at present, of a continuous-current four-pole 30-h.p. shunt-motor direct-connected to a six-pole three-phase generator of 15-kw. capacity generating-current at 1,100 volts; the speed of the set is 975 rev. per minute.

An additional transforming-group, to be installed shortly, consists of a 50-kw. separately-excited inverted rotary-converter connected to three step-up transformers. The transmission-line voltage is 1,200. The engine-house is spanned by a 20-ton traveling-crane.

The steam for the steam-engines is supplied by a set of three multitubular Tosi boilers heated by purified blast-furnace gas. Each boiler has a heating-surface of 2,850 sq. feet.

V. Pumping-Plant.

The pumping-plant is of two different sections; the cooling-water section and the feed-water section.

The cooling-water plant is situated on the sea-shore at the works, and contains four groups, each consisting of a Westinghouse 60-h.p. four-pole shunt-motor direct-connected to two Dumont centrifugal tandem-connected pumps. Motors and pumps are placed on separate bed-plates. The output of each group at a speed of 890 rev. per min. is 8,750 cu. ft. per hr.; the water is raised to a height of 99 ft. above the sea-level and stored in two metallic tanks of 8,750 cu. ft. capacity.

The sea-water is used for bosh- and tuyere-cooling and for the cylinders of the gas-engines. For cooling the ignition-chambers of the latter, the same water is supplied at a slightly higher pressure by two small steam-pumps drawing water from the discharge-pipe of the tanks.

The section of the pumping-plant which supplies feed-water to the boilers is situated at a distance of about 2 miles from the works. The water is obtained from three wells about 50 ft. deep. A reciprocating-pump driven by a high-tension induction-motor raises the water and forces it to a tank of 19,250 cu. ft. capacity placed 230 ft. above the pumps. This tank is built

partly of stone and partly hewn in the rock and from it the water runs directly to the works.

A large pumping-group, corresponding to the increased needs of the works, now being erected, will shortly be put in operation. It consists of a Westinghouse induction-motor of 40-h.p. capacity running at a speed of 720 rev. per min. with a tension of 1,000 volts. This motor is belt-connected to a Bopp & Reuther horizontal pump having three plungers, and capable of raising 2,450 cu. ft. of water per hr., the dimensions of the pump are:—Diameter of plungers, 5.1 in.; the stroke is 7.9 in., and the speed is 156 rev. per minute. There is also a reserve steam-pump to which steam is supplied by a Cornwall boiler of 240 sq. ft. heating-surface.

VI. Railways.

In addition to the above-described machinery, etc., there are two small railway-lines within the works. One, a 47-in. gauge track, serves to carry the slag-ladles to and from the furnaces. The slag-ladles are hauled by an electric-trolley locomotive equipped with two 13-h.p., 220-volt series-motors. The second railway is a steam-line of 31.5-in. gauge on which two locomotives haul the limestone and pig-iron cars to and from the blast-furnaces.

Roasting and Magnetic Separation of a Blende-Marcasite Concentrate.

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(Lake Superior Meeting, September, 1904.)

I. Introduction.

Zinc smelters in the central western States have established a very high standard of purity for blende-concentrates, viz., zinc 60, iron less than 3, and lead less than 1 per cent. The very low percentage of iron is made possible by the blende containing less than 0.5 per cent. of isomorphous iron sulphide and by the sparing occurrence of pyritic material in the large blende-deposits of southwestern Missouri and southeastern Kansas. The blende in these districts occurs in sub-Carboniferous limestone, and is associated with chert, residual clay, calcite, pyrite or marcasite and bitumen. The different minerals are coarse and therefore readily separated by mechanical washing from the blende, with the exception of pyrite and marcasite, as blende has a specific gravity of from 3.9 to 4.1; marcasite, from 4.85 to 4.90; and pyrite, from 4.95 to 5.10. The common procedure of dressing the ores[1] is to crush in a breaker, size in a trommel with 0.5-in. holes, pass the over-size through roughing-rolls and then jig the ore, now reduced to 0.5 in. maximum size, in roughing multiple-sieve jigs. The products of these are waste-tailings, side-discharge coarse concentrates, side-discharge coarse middlings and bottom-discharge fine middlings. The coarse concentrates are sent to the market. The coarse middlings are crushed in middlings-rolls and brought back into the system by feeding into the main trommel. The fine middlings are cleaned upon finishing-jigs. Any pyrite or marcasite present follows the blende and increases the percent-

[1] *Ore-Dressing*, by R. H. Richards, New York, p. 900 (1903).

age of iron above permissible limits. Several processes[2] based upon roasting for porosity (Schullsburg, Wis.; Åmmeberg, Sweden), heating for decrepitation (Heusschen Process), or disintegration followed by screening (Lintorf, Prussia), have been used for separating blende and pyrite, but they have all been, or will be, replaced by magnetic separation. Pyrite, marcasite and blende are diamagnetic, blende may be feebly paramagnetic, the attraction depending upon the percentage of isomorphous iron sulphide it contains. In order to separate pyrite or marcasite from blende running low in iron, the former has to be made magnetic. This can be done in two ways, by converting the disulphide of iron into magnetic oxide or into magnetic sulphide. In the work to be discussed both methods were tried.

II. The Ore and the Outline of the Experimental Work.

The ore used in the experiments was a coarse concentrate of about 0.5-in. size from the mines of the Alexandra Mining Co., Carterville, Mo., furnished by the superintendent, Mr. T. F. Lennan. It consisted of light-colored resinous blende (rosin jack), marcasite and chert. The chemical analysis gave Zn, 35.94, and Fe, 18.77 per cent. Calculating all the iron as marcasite, the rational analysis would be,—blende 53.57, marcasite 40.22 and gangue 6.21 per cent.

Preliminary experiments having shown that the ore as received, i.e., 0.5-in. material, was too coarse for the proposed work, an 8-mesh screen was chosen for the limiting size, as this appeared to work well and as it is the one best suited for roasting blende. In order to study the behavior of ore of smaller size, a 20-mesh screen was selected as the limit, because in crushing the 0.5-in. ore through an 8-in. sieve a very large quantity was found to be fine enough to pass through a 20-mesh screen. A series of experiments was then made on those two sizes. The operations were: 1. Crushing to required size; 2. Roasting above the ignition-point of marcasite, but below that of blende, and magnetic separation; 3. Roasting as under 2, sizing and magnetic separation of sized product; 4.

[2] *Production and Properties of Zinc*, by W. R. Ingalls, New York, p. 255 (1902). *Ore-Dressing*, by R. H. Richards, pp. 790, 880.

Sizing crushed ore, roasting and magnetic separation of sized material; 5. Analytical determination of zinc and iron.

III. Apparatus and Methods of Analysis.

The apparatus used in the experiments were two heating-furnaces with Le Chatelier pyrometers and two electro-magnets.

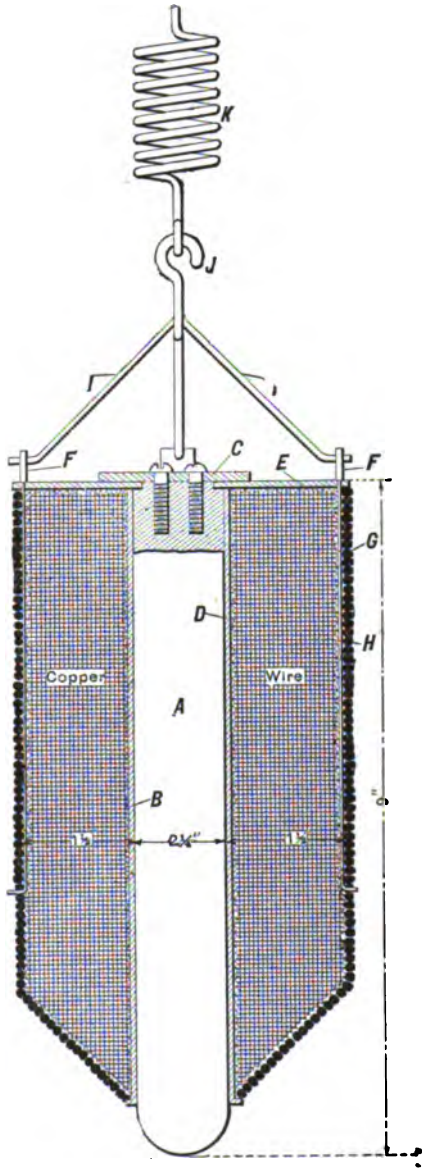
An electric heating-furnace, similar to the retort furnace described by Hofman[3] in a paper read at the Lake Superior meeting of the Institute, September, 1904, served to determine the ignition-points of marcasite and blende. The roasting-operations were carried on in 5-in. roasting-dishes heated in a gas muffle-furnace, made by the American Gas Furnace Co., New York, with a muffle 12 in. long, 6.75 in. wide and 3.75 in. high. While one charge was being heated in the muffle, a dish charged with ore was warmed at the front so that it might be brought quickly to the muffle-temperature when the ore was to be roasted.

In the measurement of temperature the two leads of the Le Chatelier pyrometers, wound with asbestos thread, were held in a porcelain tube. The thermo-couple protruding at one end was about at the level of the ore-charge when the porcelain tube was lying on the floor of the muffle close to the roasting-dish. In this position the temperatures were taken. When the muffle had been brought to the desired temperature, it required very little regulating of the gas- and air-supply to hold it there.

For the magnetic separation of the roasted ore, two electro-magnets constructed for laboratory purposes, by Prof. R. H. Richards, were used, one, a single-pole; the other, a double-pole. The single-pole magnet is shown in Fig. 1. The core, *A*, a cylinder of soft Norway iron, fits into a brass tube, *B*, carrying the coils of copper wire; it is rounded at one end and screwed to a brass disk, *C*, at the other. It is enclosed by three layers of paper, *D*. There are 9,240 turns of double cotton-covered No. 21 copper wire, equal to a length of 8,100 feet. At the top the coil is held in place by the brass disk and protected by three rings of paper, *E*; at the bottom tape is wound in with the wire to prevent the coils from slipping. In winding the

[3] The Decomposition and Formation of Zinc Sulphate by Heating and Roasting, see p. 815.

FIG. 1.



SINGLE-POLE ELECTRO-MAGNET.

wire downward, four pieces of linen tape, 4 in. long and 0.25 in. wide, are placed for a distance of 2 in. on the sides of the core. When covered with wire, the tape was bent upon itself, and the free ends now covered with wire in winding upward.

Thus, each winding is held in place near the bottom of the core by four loops of tape and prevented from slipping. The winding is such that a current of 0.8 amperes at a pressure of 50 volts gives the maximum of magnetic lines the core can carry without undue heating. The magnet is suspended by four strips of brass, *F*, held in place by $\frac{3}{16}$-in. twine (tarred marline), *G*, wound closely around the coils previously covered with three layers of paper, *H*. The twine forms at the same time a protective casing for the magnet. Each strip has an eye near the upper end, a stout wire, *I*, is passed through it, the four wires are wound together and bent to a hook, *J*, and hung on a steel spring, *K*, suspended by an iron rod from the ceiling.

The double-pole magnet, shown in Fig. 2, has two cores of soft iron joined at the top by a yoke of bar-iron; the pole-shoes, with faces 1 by $\frac{1}{8}$ in., are screwed to the bottoms of the cores, leaving an air-gap $\frac{5}{8}$ in. long between the north and south poles. The magnet is suspended by a screw-eye from a spring in the same manner as the single-pole magnet. Each core is wound with 5,000 ft. of the same wire as the single-pole magnet, making 6,760 coils. The turns are held in place at top and bottom by brass rings and are protected by layers of paper and tarred marline. The saturation-point is the same as that of the single-pole magnet.

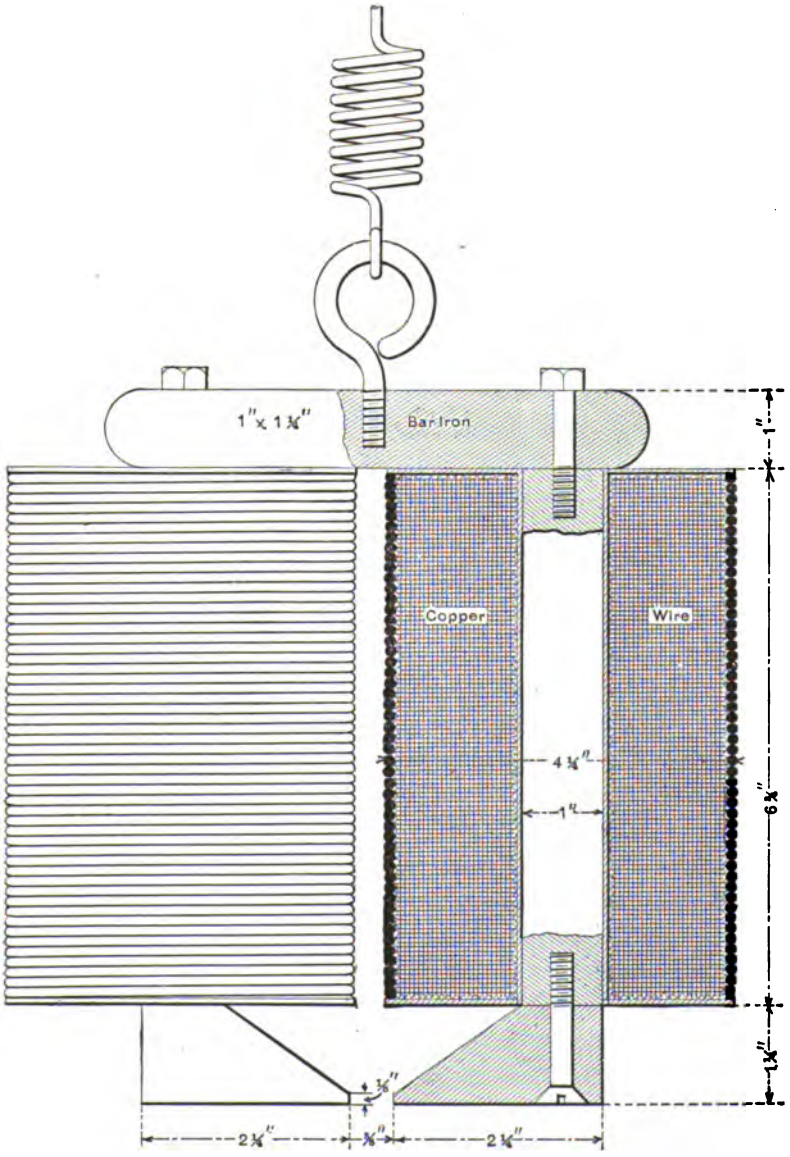
The attractive power of a magnet varies with the number of lines of force that pass through it. With the single-pole magnet there is the greatest scattering of lines of force, with the double-pole magnet the greatest concentration, hence for a given electric current the attraction per sq. cm. of field in a single-pole magnet is much smaller, about 60 per cent., than in the double-pole magnet, as long as the air-gap between the two poles is not too large.

The current for the magnets is taken by a shunt from the 110-volt electric-light main. In order to reduce it to the required strength, an incandescent lamp of eight candle power strength is placed in the circuit. From the magnet the current passes through a Weston direct-reading ammeter back to the main. A switch is placed in the circuit to permit the current to be turned on and off.

In a test the ore was spread out on a piece of glazed paper, the current turned on, the single-pole magnet passed to and

fro over the ore with raising and lowering, and then moved to one side, the current switched off and the adhering ore

FIG. 2.



DOUBLE-POLE ELECTRO-MAGNET.

dropped onto a paper. Another layer of ore was then spread over the first glazed paper, and the operations repeated until a

batch had been treated. The first magnetic concentrates were then retreated until they had been freed from the non-magnetic particles that had been carried along mechanically. As a rule the tailings from the single-pole magnet were treated with the double-pole magnet, the manipulation being the same as in the first operation.

The zinc in the ore and the tailings was determined by titrating with potassium ferrocyanide as worked out by von Schulz and Low in 1900.[4] With the magnetic concentrates, which run high in iron and are not decomposed by treatment with nitric acid and potassium chlorate, solution with nitro-hydrochloric acid had to be substituted. The iron was converted into sulphate and titrated with potassium permanganate.

IV. Experiment with 8-Mesh Material.

1. *Crushing.*—The ore as received, of 0.5-in. size, was crushed in a Hendrie and Bolthoff sample-grinder to pass through an 8-mesh sieve. The screen-analysis of the product is given in Table I.

Table I.—*Screen-Analysis of 8-Mesh Ore.*

Size of Screen.	Grams.	Per Cent.
Through 8- and on 12-mesh.....	54	33.1
Through 12- and on 16-mesh.....	18	11.0
Through 16- and on 20-mesh.....	16	9.8
Through 20-mesh.....	75	46.0
Totals.....	163	99.9

The data in Table I. show that 46 per cent. of the ground ore was fine enough to pass through a 20-mesh sieve. It is this quantity which decided the choice of 20-mesh material for the tests on fine ore.

2. *Roasting and Magnetic Separation of Unsized Material.*—Before any roasting could be undertaken, it was necessary to determine the limits of temperature which had to lie above the ignition-point of marcasite and below that of blende. The two points were found to be 378° C. for marcasite and 600° C. for blende. The first tests were planned to keep the time constant

[4] *Production and Properties of Zinc*, by W. R. Igalls, p. 99.

and to vary the temperature. The leading facts and results are brought together in Table II.

TABLE II.—*Experiments with Unsized 8-Mesh Material.*

Roasting.								Magnetic Separation.								
									Heads.					Tailings.		
									Single-Pole.			Double-Pole.				
No. of Sample.	Raw Ore.	Depth of Charge at Center.	Roasted Ore.	Loss in Roast.	Loss in Roast.	Temperature of Muffle.	Duration of Roast.	Weight Taken.		On Weight Taken.	On Raw Ore.		On Weight Taken.	On Raw Ore.		On Weight Taken.	On Raw Ore.
	Grams.	Inches.	Grams.	Grams.	Per Cent.	Deg. C.	Min.	Grams.	Grams.	Per Cent.	Per Cent.	Grams.	Per Cent.	Per Cent.	Grams.	Per Cent.	Per Cent.
1	200		175	25	14.3	400	105	175	.....	.....	.....	4	2.2	2.0	171	97.8	85.5
2	200		168	32	19.0	410	110	168	.....	.....	.....	5	2.9	2.5	163	97.1	81.5
3	200		170	30	17.7	460	120	170	.....	.....	.....	19	11.2	9.5	151	88.8	75.5
4	200		165	35	21.3	490	125	165	7	4.2	3.5	16	9.7	8.0	142	86.1	71.0
5	200		159	41	25.9	560	120	159	12	7.5	6.0	20	6.3	10.0	127	86.2	63.5
6	200		158	42	26.6	620	120	158	17	10.8	8.5	19	12.0	9.5	122	77.2	61.0

In the six tests made to convert by long-time roasts marcasite into magnetic iron oxide, the time given to roasts Nos. 3, 5 and 6 was 120 min.; roast No. 4 was accidently continued 5 min. longer than the other three; roasts Nos. 1 and 2, serving as feelers, lasted only 105 and 110 min. respectively. The temperatures show a range of from 400 to 620° C. The rise above 600° C. was due to the heat generated in the burning of marcasite; it caused some blende to become superficially oxidized. The concentration-tests show that at a temperature ranging from 400 to 460° C., the ore is not sufficiently altered to be attracted by the single-pole magnet. With a rise of temperature the extraction begins with 3.5 per cent. and increases to the moderate figure of 8.5 per cent. when the limit of temperature set has been overstepped.

The double-pole magnet, within the range of from 400 to 460° C., draws out from 2.0 to 9.5 per cent. of magnetic material which was not acted upon by the single-pole. The higher temperatures, 460° and 620° C., give an additional yield of from 8 to 10 per cent. to that obtained by the single-pole magnet, but at best the total extraction of 18 per cent. is very unsatisfactory. Most of the ore removed by the magnet was fine;

coarse particles could be moved about on the paper, but not made to adhere to the magnet. The results ought, therefore, to be improved by fine-crushing. The screen-analyses of the roasted products given in Table III. show that the 46 per cent. of material finer than 20-mesh size in the raw ore has been increased to about 60 per cent. in the roasted ore. This must be attributed to decrepitation. The 3 or 4 per cent. of ore which remained on the 8-mesh sieve find their explanation in the swelling of particles during the roast.

TABLE III.—*Screen-Analysis of Roasted 8-Mesh Material.*

No. of Sample.	Weight Taken.	On 8-Mesh.		On 12-Mesh.		On 16-Mesh.		On 20-Mesh.		Through 20-Mesh.	
	Grams.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.
1	168	7	4.1	23	13.7	16	9.5	15	8.9	107	63.7
2	160	6	3.7	27	16.9	14	8.7	15	9.3	98	61.2
3	158	7	4.4	27	17.1	14	8.8	14	8.8	96	60.8
4	154	7	4.5	23	14.9	13	8.4	14	9.0	97	63.0
5	152	6	3.9	33	21.7	15	9.8	16	10.5	82	53.9
6	161	7	4.3	29	18.0	12	7.4	14	8.7	98	60.8

V. EXPERIMENTS WITH 20-MESH MATERIAL; UNSIZED.

1. *Crushing.*—The 8-mesh material was ground to pass a 20-mesh screen in the same manner as the original ore. The sizes obtained in the operation are given in Table IV. It shows

TABLE IV.—*Screen-Analysis of 20-Mesh Ore.*

Size of Screen.	Grams.	Per Cent.
Through 20- and on 40-mesh.....	89	44.5
Through 40- and on 60-mesh.....	34	17.0
Through 60- and on 80-mesh.....	10	5.0
Through 80-mesh.....	67	33.5
Total.....	200	100.0

that of the ore 45.5 per cent. was coarser than a 40-mesh, while 33.5 per cent. passed through an 80-mesh screen.

2. *Roasting and Magnetic Separation of Unsized Ore.*—As the results in Table II. had pointed to the conclusion that a higher temperature was more favorable to magnetization than a lower

one, the former was chosen. The roasts were carried on at the constant temperature of 600° C., and the time gradually reduced from 120 to 1.25 minutes. The weight of a charge was 100 g., which gave a layer 0.5-in. deep in the center of the roasting-dish. In the roasts lasting longer than 5 min., the temperature-readings were taken every 5 min., and the ore was stirred every 2 min.; in the roasts lasting 5 min. or less, the temperature was read at the beginning and the end of the roast, and the ore stirred continuously. The appearance of the roasted ore varied with the time it had been exposed to heating; that from the long-time roasts was brownish, due to the more perfect oxidation; that from the short-time roasts black, on account of the partial conversion of iron disulphide into magnetic monosulphide.

After the completion of a roast, the ore was weighed, a sample taken for chemical analysis and the rest subjected to the single-pole and then to the double-pole magnet. A small portion of the zinc contained in the magnetic heads from the single-pole and especially from the double-pole magnet was not carried along mechanically, but was itself attracted by the magnet, showing that the very small quantity of iron sulphide contained in the light-brown resinous blende was sufficient to cause this to show some magnetic property. The results are assembled in Table V., which is best made clear by following through one test. In test No. 9, *e.g.*, 100 g. of raw ore, spread over the roasting-dish, gave a layer 0.5 in. thick in the center of the dish. The ore was roasted for 1.25 min. at a temperature of 600° C. The roasted ore weighed 95 g., thus losing 5 per cent. in weight by the operation. From the 95 g. roasted ore, 15 g. were reserved for chemical analysis, leaving 80 g. for magnetic treatment.

Table V., however, gives only the data calculated to the basis of the total weight of the roasted ore, i.e., 95 grams. The single-pole magnet took out 23.7 g., the double-pole magnet 15.5 g., giving a total of 39.2 g. of magnetic material, and leaving 55.8 g. of tailings. The percentage of zinc of the single-pole product was found by calculation: The total quantity of zinc present in the raw or roasted ore was 35.94 grams. The tailings gave by analysis 55.98 per cent. of zinc; they contained therefore $0.558 \times 55.9 = 31.19$ g. of zinc; the double-pole

TABLE V.—*Experiments with 20-Mesh Material (35.94 Per Cent. of Zn), Unsized.*

Roasting.							Magnetic Separation.														
							Heads.										Tailings.				
							Single-Pole.				Double-Pole.				Total.						
No. of Sample.	Raw Ore.	Depth of Charge.	Roasted Ore.	Loss of Weight in Roast.	Temperature of Muffle.	Duration of Roast.	Calculated on Weight of Roasted Ore.	Zinc, Calculated.	Weight, Zinc.	Of the Zinc in the Raw Ore.	Calculated on Weight of Roasted Ore.	Zinc by Analysis.	Weight, Zinc.	Of the Zinc in the Raw Ore.	Calculated on Weight of Roasted Ore.	Weight of Zinc.	Of the Zinc in the Raw Ore.	Calculated on Weight of Roasted Ore.	Zinc by Analysis.	Weight of Zinc.	Of the Zinc in the Raw Ore.	Iron by Analysis.
	Grams.	Inches.	Grams.	Per Cent.	Deg. C.	Min.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Per Cent.
1	100	0.5	78	22	600	120	0	0	0	0	32.9	11.04	3.63	10.10	32.9	3.63	10.10	45.1	71.17	32.10	89.23	3.78
2	100	0.5	81	19	600	80	0	0	0	0	31.3	2.36	0.74	2.05	31.3	0.74	2.05	50.1	66.94	33.54	93.30	2.78
3	100	0.5	83	17	600	60	4.3	0	0	0	26.9	7.20	1.94	5.40	31.2	1.94	5.40	51.8	65.69	34.02	94.64	2.81
4	100	0.5	86	14	600	30	15.6	0	0	0	20.1	19.58	3.94	10.96	35.7	3.94	10.96	50.3	64.52	32.45	90.28	1.57
5	100	0.5	87	13	600	15	20.9	0	0	0	15.4	29.16	4.49	12.49	36.3	4.49	12.49	50.7	64.36	32.63	90.78	1.32
6	100	0.5	89	11	600	10	29.7	1.91	0.67	1.86	10.4	37.41	4.26	11.86	40.1	4.93	13.72	47.9	64.76	31.01	86.28	0.90
7	100	0.5	89	11	600	5	30.4	2.86	0.87	2.42	10.2	35.24	3.56	9.90	40.6	4.43	12.32	48.5	64.97	31.51	87.68	0.90
8	100	0.5	92	8	600	2.5	23.8	2.27	0.54	1.50	14.8	26.73	3.93	10.94	38.6	4.47	12.44	53.5	58.53	31.47	87.56	5.13
9	100	0.5	95	5	600	1.25	23.7	2.57	0.61	1.69	15.5	26.29	4.04	11.25	39.2	4.65	12.94	55.8	55.98	31.19	87.06	7.05

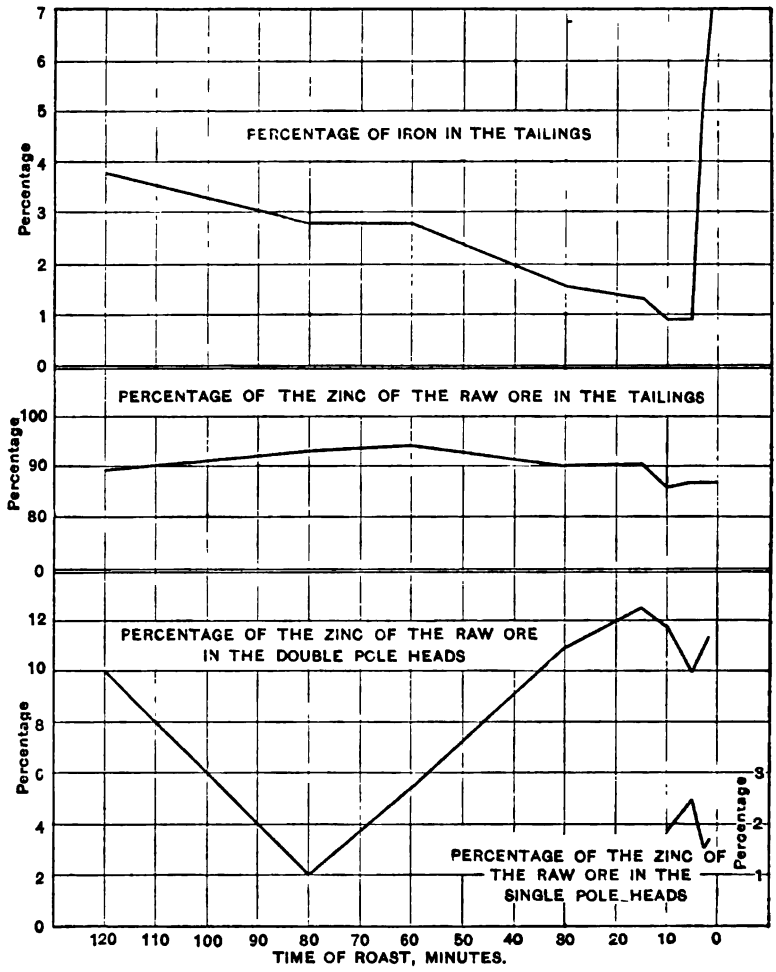
magnetic product gave 26.29 per cent. of zinc and contained $0.2629 \times 15.5 = 4.04$ g. of zinc; the single-pole concentrates therefore contained $35.94 - (31.19 + 4.04) = 0.71$ g., which corresponds to $\frac{0.71}{23.7} \times 100 = 3.00$ per cent. of zinc. The distribution of the zinc in the raw ore among the three products was next calculated and found to be, in the single-pole product, $\frac{0.71}{35.94} \times 100 = 1.97$ per cent.; in the double-pole product, $\frac{4.04}{35.94} \times 100 = 11.25$ per cent., or a total of 12.94 per cent.; in the tailings, $\frac{31.19}{35.94} \times 100 = 86.78$ per cent. Thus 86.78 per cent. of the total quantity of zinc was recovered in the tailings which by analysis gave 7.05 per cent. of iron.

Reviewing the data in Table V. as a whole, it is seen that when the ore is roasted from 80 to 120 min. nothing is extracted by the single-pole magnet. As the time of roasting decreases, the ore becomes more and more magnetic until with a 5-min. roast a maximum of 30.4 g. is obtained. With roasts of shorter duration the quantity extracted decreases again. Using the double-pole after the single-pole magnet, 32.9 g. are taken out with the 120-min. roast, the quantity removed diminishes with the time of the roast, until with a 5-min. roast the minimum is reached with 10.2 grams. Reducing the time below 5 min. again increases the quantity of concentrate. The results of 10-min. roasts differ little from those obtained in roasting 5 minutes.

As far then as quantity of material extracted is concerned, a roast ranging from 5 to 10 min. gives the best results, as during this period the ore becomes sufficiently magnetic to allow a maximum to be removed by the single-pole magnet or by a double-pole magnet of lower magneto-motive force. The quality of the products, i.e., the percentage of zinc remaining in the concentrates, which is lost, and in the tailings, which is recovered; and the percentage of iron retained by the tailings, are plotted in Fig. 3 from the data in the table. The abscissa gives the duration of the roasts in minutes; on the ordinates are inscribed the percentages in four different scales to permit bringing the whole together in a single sheet and thus facilitate comparison. In roasting the ore 120 min., the single-pole magnet

took out nothing, the double-pole concentrates carried along with them 10.10 per cent. of the zinc contained in the ore, while in the tailings there was recovered 89.23 per cent.; the tailings assayed 3.78 per cent. of iron. The quantity of zinc

FIG. 3.



DIAGRAMMATIC SKETCH OF DATA OF TABLE V.

retained by the double-pole magnet decreased to 2.05 per cent. with the shortening of the time of roasting to 80 min.; it then again increased until a maximum of 12.49 per cent. was reached with 15 min.; this was followed by a second minimum of 9.90 per cent., with 5 minutes. The single-pole heads did not retain

any zinc until the time of roasting had been reduced to 10 min., when they held 1.86 per cent. of the total quantity of zinc; at 5 min. their zinc-content reached a maximum of 2.42 per cent., which is followed by a minimum of 1.5 per cent. at 2.5 minutes.

The tailings show an approximately corresponding variation in zinc, ranging from 86.23 to 94.64 per cent., and a steadily diminishing iron-content until the 5-min. roast has been reached with 0.9 per cent. of iron, when there is a sudden increase. The most favorable result for the single-pole magnet, 1.50 per cent. of the total quantity of zinc, is obtained with a roast lasting 2.5 min.; with the double-pole magnet 80 min. give the minimum of zinc, 2.05 per cent.; the tailings furnish the highest yield in zinc, 94.64 per cent., when the roast is prolonged for 60 min., but unfortunately the percentage of iron, 2.81 per cent., is high.

Combining the four points of view, the best quality of product is obtained in roasting from 5 to 10 min.; the single-pole heads retain about 2.25 per cent. of the total quantity of zinc, the double-pole heads about 11 per cent. and the tailings collect from 86 to 87 per cent. The raw ore, with 35.94 per cent. of zinc and 18.77 per cent. of iron, has been enriched to a market blende assaying 64.8 per cent. of zinc and 0.9 per cent. of iron; and the yield of zinc is from 86 to 87 per cent.

A screen-analysis of the roasted ore, of the magnetic concentrates and of the tailings, was carried through in order to study the changes in size, etc., that the ore (see Table IV.) and tailings had undergone in roasting, and to find what special sizes had been attracted by the magnets. The results are given in Table VI., in which Nos. 1 to 9 correspond to the similar designations in Table V. Comparing Tables IV. and VI., it is seen that about 3 per cent. of the ore, during the roast, increased sufficiently in size so as not to pass through a 20-mesh screen. While with the raw ore, 45.5 per cent. was caught on a 40-mesh sieve, the roasted ore yielded only 35 per cent.; the material remaining on a 60-mesh and passing through an 80-mesh sieve is about the same in both cases, 17 and 33 per cent., while that held by the 80-mesh screen is smaller with the raw ore (5 per cent.) than with the roasted ore (9 per cent.). These changes are due to the decrepitation of the blende. Of the concentrates taken out by the single-pole magnet, about 45 per

TABLE VI.—*Screen-Analysis of Roasted, Unsized 20-Mesh Material.*

No. of Sample.	Roasted Ore.										Magnetic Heads.														Tailings.									
											Single-Pole.										Double-Pole.														
	Weight Taken.	On 20.		On 40.		On 60.		On 80.		Through 80.	Weight Taken.	On 40.		On 60.		On 80.		Through 80.			Weight Taken.	On 40.		On 60.		On 80.		Through 80.			Weight Taken.	On 40.		On 60.		On 80.		Through 80.	
	Grams.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Grams.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.		Grams.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.
1	71	2	3	24	34	14	19	7	9	24	34	.....	.....	.....	.....	.....	.....	.....	.....	.....	30	9	30	5	16	3	10	13	43	41	14	34	8	19	5	12	14	34
2	75	3	4	26	34	14	18	7	9	25	34	.....	.....	.....	.....	.....	.....	.....	.....	.....	29	11	37	4	13	2	7	12	41	46	19	41	7	15	4	8	16	34
3	77	2	3	27	35	14	18	7	9	27	35	4	.....	.....	.....	.....	.....	.....	.....	.....	25	10	40	4	16	2	8	9	36	48	17	35	10	21	5	10	16	33
4	77	4	5	27	35	16	20	7	9	23	30	14	5	36	1	7	2	12	6	43	18	7	38	3	16	1	6	7	39	45	18	35	7	15	5	11	15	33
5	79	2	2	26	33	16	20	7	9	28	34	19	9	46	4	21	1	5	5	26	14	5	35	1	7	1	7	7	50	46	17	37	7	15	5	11	17	37
6	78	3	3	29	36	15	19	6	8	25	32	26	12	46	4	15	1	4	8	30	10	3	30	2	20	.....	.....	5	50	42	15	35	9	21	4	9	17	40
7	79	3	3	29	35	15	19	5	8	28	33	27	11	41	5	14	1	4	9	33	9	3	33	1	11	1	11	4	44	43	15	35	10	23	5	11	13	30
8	81	2	2	28	34	14	17	8	7	29	35	21	10	47	3	14	1	5	7	33	13	4	30	1	7	1	7	7	53	44	18	40	9	20	5	11	12	28
9	80	2	2	29	36	15	18	7	9	26	32	20	8	40	4	20	1	5	6	30	13	4	30	1	7	1	7	6	46	46	19	41	11	23	4	9	12	27

cent. is coarser than a 40-mesh and about 35 per cent. finer than an 80-mesh sieve. With the double-pole magnet, a little less than 35 per cent. is coarser than a 40-mesh and over 45 per cent. finer than an 80-mesh sieve. The tailings give similar figures for the larger-size material; about 35 per cent. will be caught by a 40-mesh screen, while a smaller amount of fines, 33 per cent., passes through an 80-mesh screen.

VI. Experiments with 20-Mesh Material; Roasting, Sizing and Magnetic Separation.

The aim of this series of tests was to find out whether a better magnetic separation could be obtained with sized than with unsized material, as in the preceding experiments. Some of the experiments on 8-mesh unsized material (Table I.) had shown that, while the fine particles were readily attracted by the magnet, the coarser ones could only be made to travel over the paper when the magnet was moved to and fro.

1. *Crushing.*—The ore used in the tests was taken from the same lot as that in the preceding work.

2. *Roasting, Sizing and Magnetic Separation.*—In order to have enough material of the different sizes for magnetic treatment, it was necessary to use 300 grams. The sample was, however, roasted in three separate portions of 100 g. each, in order to have the same conditions as before as far as quantity was concerned. The temperature of the muffle was maintained at 600° C., and the time given to a roast reduced to 5 min., as the previous work had shown that this gave the best results. The loss in weight in roasting was 8 per cent.

Sizing the roasted ore gave the subjoined data.

Table VII.—*Screen-Analysis of 20-Mesh Material, Roasted.*

Size of Screen.	Grams.	Per Cent.
Through 20- and on 40-mesh.....	99.3	36
Through 40- and on 60-mesh.....	53.6	19
Through 60- and on 80-mesh.....	25.5	8
Through 80-mesh.....	96.8	35
Totals.....	275.2	98

The results are similar to those of Table VI. Of the ore 36

per cent. is coarser than a 40-mesh and 35 per cent. finer than an 80-mesh screen.

The sized material was subjected first to the single-pole and then to the double-pole magnet as described in the preceding pages. The results obtained are assembled in Table VIII., which is put together on the same general lines as Table V., with the exception that the percentage of zinc of each of the three products had to be determined analytically, as any calculation was excluded by the circumstance that the total zinc present in a mesh-size was an unknown quantity.

An examination of Table VIII. shows that a good separation was obtained with material too coarse to pass an 80-mesh screen. Thus, 40-mesh ore, which comprised 36 per cent. of the ore treated, gave tailings running 61.70 per cent. of zinc and 2.02 per cent. of iron with a loss of only 5.02 per cent. of the total quantity of zinc. Material of 60- and 80-mesh sizes gave tailings of a similar tenor in zinc with an even smaller loss in total zinc. The work upon ore finer than an 80-mesh screen gave unsatisfactory results. Forming by weight 35 per cent. of the ore subjected to magnetic treatment, the tailings could not be brought to run higher than 54.74 per cent. of zinc and lower than 5.42 per cent. of iron, accompanied by a loss in the concentrates of 10.18 per cent. of the zinc present.

The sample of 300 g. of raw ore contained (3×35.94), = 106.92 g. of zinc. Of this, 80.76 per cent. was recovered in the tailings and 19.22 per cent. lost in the concentrates, 5.16 per cent. in those of the single-pole, and 14.06 per cent. of the double-pole, the main loss being due to material finer than an 80-mesh sieve. The results are inferior to those obtained with unsized roasted ore. The plan of sizing roasted ore before treatment with the magnet is not advantageous.

VII. Experiments with 20-Mesh Material; Sizing, Roasting and Magnetic Separation.

The experiment still remaining to be made to complete the series is to size the 20-mesh ore, roast the sized products severally and then subject them to magnetic treatment.

1. *Crushing.*—The ore used in the work was taken from the main lot which had been passed through a 20-mesh screen.

TABLE VIII.—*Experiments with 20-Mesh Material; Roasted, Sized and Separated.*

Weight from Table VII.	Magnetic Separation.																											
	Mesh-Size.	Weight Taken.	Heads.																Tailings.									
			Single-Pole.						Double-Pole.						Total.													
				On Weight Taken.	On Raw Ore.	Zinc.	Weight, Zinc.	Of the Zinc in the Raw Ore.		On Weight Taken.	On Raw Ore.	Zinc.	Weight, Zinc.	Of the Zinc in the Raw Ore.		On Weight Taken.	On Raw Ore.	Weight Zinc.	Of the Zinc in the Raw Ore.		On Weight Taken.	On Raw Ore.	Zinc.	Weight, Zinc.	Of the Zinc in the Raw Ore.	Iron.	
Per Cent.		Grams.	Grams.	Per Cent.	Per Cent.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Per Cent.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Per Cent.	Per Cent.	Grams.	Per Cent.	Per Cent.	
36	On 40.	99.3	30.6	30.8	10.2	3.58	1.10	1.02	15.3	15.4	5.1	28.15	4.31	4.00	45.9	46.2	15.3	5.41	5.02	68.4	68.8	17.8	61.70	32.89	30.52	2.02
19	On 60.	53.6	12.1	22.5	4.0	3.90	0.47	0.44	7.8	14.5	2.6	28.84	2.83	2.62	19.9	37.0	6.6	3.30	3.06	33.7	63.0	11.2	60.68	20.45	18.97	2.44
8	On 80.	25.5	5.1	20.0	1.7	3.32	0.17	0.16	3.6	14.1	1.2	23.91	0.86	0.80	8.7	34.1	2.9	1.03	0.96	16.8	65.9	5.6	60.91	10.23	9.49	2.49
35	Through 80.	96.8	31.1	31.5	10.4	12.29	3.82	3.54	22.8	23.5	7.6	31.39	7.16	6.64	53.9	55.0	17.9	10.98	10.18	42.9	45.0	14.3	54.74	23.48	21.78	5.42
Totals..								5.16						14.06					19.22						80.76	

NOTE.—The total quantity of zinc in raw ore equals (3×35.94), 106.92 grams.

TABLE IX.—*Experiments with 20-Mesh Material; Sized, Roasted and Separated.*

Roasting.							Magnetic Separation.																									
								Heads.																	Tailings.							
								Single-Pole.						Double-Pole.						Total.												
Size of Screen.	Raw Ore.	Depth of Charge.	Roasted Ore.	Loss in Roast.	Temperature of Muffle.	Duration of Roast.	Weight Taken.		On Weight Taken.	On Raw Ore.	Zinc.	Weight, Zinc.	Of the Zinc in the Raw Ore.		On Weight Taken.	On Raw Ore.	Zinc.	Weight, Zinc.	Of the Zinc in the Raw Ore.		On Weight Taken.	On Raw Ore.	Weight, Zinc.	Of the Zinc in the Raw Ore.		On Weight Taken.	On Raw Ore.	Zinc.	Weight, Zinc.	Of the Zinc in the Raw Ore.	Iron.	Weight Zinc in Raw Ore, *i.e.*, in Heads and Tails.
	Grams.	Inches.	Grams.	Per Cent.	Deg. C.	Min.	Grams.	Grams.	Per Cent.	Per Cent.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Per Cent.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Per Cent.	Per Cent.	Grams.	Per Cent.	Per Cent.	Grams.
On 40.	100	½	92.6	7.4	600	5	92.6	28.6	30.8	28.6	1.85	0.53	1.52	8.7	9.4	8.7	28.28	2.46	7.05	37.3	40.2	37.3	2.99	8.57	55.3	59.8	55.3	57.73	31.98	91.43	2.28	34.92
On 60.	100	½	93.8	6.2	600	5	93.8	21.7	23.1	21.7	2.59	0.56	1.54	11.4	12.1	11.4	20.85	2.38	6.51	33.1	35.2	33.1	2.94	8.05	60.7	64.8	60.7	55.35	33.57	91.95	3.30	36.51
On 80.	100	½	93.9	6.1	600	5	93.9	20.6	21.9	20.6	2.75	0.56	1.49	10.8	11.5	10.8	18.80	2.08	5.34	31.4	33.4	31.4	2.59	6.83	62.5	66.6	62.5	56.67	35.42	93.17	2.65	38.01
Through 80.	100	½	91.3	8.7	600	5	91.3	33.1	36.2	33.1	12.07	3.99	14.22	18.4	20.1	18.4	29.36	5.40	19.24	51.5	56.8	51.5	9.39	33.46	39.8	43.7	39.8	46.95	18.68	66.54	8.12	28.07

2. *Sizing, Roasting and Magnetic Separation.*—From the screen-analysis given in Table IV. it is seen that, in order to obtain 100 g. of ore passing through a 60- and remaining on an 80-mesh sieve, 2,000 g. have to be taken for a test. The unit of ore subjected to roasting was 100 g., the temperature again 600° C. and the time 5 minutes. The roasted ore was first treated with the single-pole and then with the double-pole magnet. The results are given in Table IX.

In general character they are similar to those recorded in Table VIII., in which the roasting preceded the sizing, but denote an improvement. In the sized material coarser than an 80-mesh screen, about 92 per cent. of the total quantity of zinc is saved in the tailings which, however, run high in iron, from 2.65 to 3.30 per cent. The ore passing through an 80-mesh screen saves only 66.54 per cent. of the total quantity of zinc, and the tailings assay 8.12 per cent. of iron, which makes the method impracticable.

VIII. Conclusion.

The conclusion to be drawn from the experiments points in one direction only. Blende and marcasite in the concentrate under consideration can be successfully separated by crushing through a 20-mesh screen, roasting from 5 to 10 min. at a temperature not exceeding 600° C., and then subjecting to two double-pole magnets, the first having less than half the strength of the second.

The Effect of Silver on the Chlorination and Bromination of Gold.

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INTRODUCTION.

WHEN dry chlorine gas is made to act in the cold upon finely-divided gold,[1] it converts the latter with evolution of heat into auro-auric chloride, Au_2Cl_4, a hard, dark-red, hygroscopic salt. Moisture splits this salt into aurous and auric chloride, $Au_2Cl_4 = AuCl + AuCl_3$; treatment with water converts it into auric chloride and gold, $3\ Au_2Cl_4 = 4\ AuCl_3 + Au_2$. Aurous chloride, when stirred with water, undergoes a similar decomposition, $3\ AuCl = AuCl_3 + Au_2$. These decompositions of auro-auric chloride and of aurous chloride furnish the explanation for the practice of moistening an ore before it is treated by the Plattner chlorination-process; practical experience having shown that gaseous chlorine gave an unsatisfactory extraction with dry ore.

According to Rose,[2] fine gold is acted upon more slowly by chlorine than gold containing some base metal, e.g., copper. He also says that small quantities of silver increase the rate of solution, but adds that the coating of silver chloride formed, checks, and finally stops, further action, if the percentage of silver be increased beyond a certain undetermined amount.

Bromine[3] acts upon finely-divided gold in a manner similar to chlorine; the auro-auric bromide, Au_2Br_4, however, is not hygroscopic. Upon treatment with water it is decomposed as is the corresponding chlorine salt. It is generally believed that bromine does not act as energetically upon gold as does chlorine. This seems to be borne out by the results in the leach-

[1] Thomsen, *Journal für Praktische Chemie*, xiii., p. 337 (1876).
[2] *Metallurgy of Gold*, by T. K. Rose, London, p. 268 (1902).
[3] Thomsen, *Op. cit.*

ing plant of the Black Hills Milling and Smelting Co., Rapid City, S. D.,[4] where barrel-chlorination and bromination were compared in the laboratory and in the mill, and chlorine was found to make poorer tailings than bromine. Rose,[5] however, in experimenting upon the rate of solution of gold by chlorine and bromine, found that bromine dissolved gold more rapidly than chlorine, and that the action of both was quicker at from 50° to 60° C. than at ordinary temperature.

Percy[6] describes two striking lecture-experiments which show how readily silver leaf is converted into silver chloride or bromide by the action of gaseous chlorine or chlorine water, or by bromine vapor or bromine water. A large bottle is filled loosely with silver leaf; upon introducing chlorine or bromine the leaf is entirely converted in a short time into white chloride or yellowish bromide. As to the effect of silver upon chlorination of gold, the only figures published are those of Dietzsch,[7] Wagemann[8] and Coignet.[9] Dietzsch and Wagemann say that the gold must be at least 0.917 fine, if a satisfactory extraction is to be obtained, and Coignet asserts that gold with from 10 to 12 per cent. of silver is more readily dissolved than pure gold, but that gold with from 40 to 50 per cent. of silver cannot be successfully treated by chlorine with a view of dissolving the gold.

In regard to the effect of silver on bromination of gold, no data appear to have been published.

In chlorinating gold with gaseous chlorine in a stationary vat, the gold should be more effectively protected by the silver chloride formed than when chlorine-water or bromine-water is used in a revolving barrel, as the coating of silver chloride or bromide formed will be removed more or less by the abrasive action of the charge upon the metallic particles.

The aim of the present research was to find the dissolving power of chlorine-water and of bromine-water upon gold and

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[4] Godshall, *Engineering and Mining Journal*, Jan. 13, 1894, p. 32.

[5] *Metallurgy of Gold*, pp. 267, 268.

[6] *Metallurgy of Silver and Gold*, by John Percy, London, Part I., pp. 56 and 110 (1880).

[7] *Berg- und Huettenmaennische Zeitung*, xl., 13 (1881).

[8] *Berg- und Huettenmaennische Zeitung*, xlviii., 249 (1889).

[9] *Bulletin de la Société de l'Industrie Minérale*, xii., 762 (1898).

upon a series of alloys of gold and silver, the operations being carried out in revolving vessels.

The Ore-Charge.

The ore used in the experiments was made up of quartzite and gold, resp. gold-silver alloy. The quartzite was crushed to pass a 40-mesh sieve and freed from particles of iron from the crushing-machinery by means of a magnet, and by boiling with acid. The alloys were prepared from chemically pure gold and silver. The metals were rolled into strips, the desired quantities then weighed out to 0.01 mg. and alloyed by fusing on charcoal before the blow-pipe. The resulting globules were hammered to thin disks and each converted into a fine powder by holding in pincers and rubbing the edge on a fine file, using a very slight pressure. Any particles appearing to be coarse under a magnifying glass were put aside. The quantities used were too small to allow the use of a limiting sieve; the size of particles is therefore not definitely given. A charge was made up of 5 assay-tons of 40-mesh quartzite and 10 mg. of metal, making thus an ore assaying 2 oz. per ton. The composition of the series of alloys tested was—

	mg.	mg.	mg.	mg.	mg.
Gold	10	9	8	7	6
Silver	0	1	2	3	4

The Apparatus.

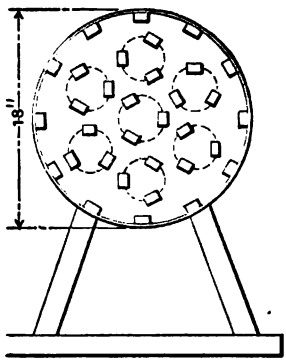
The dissolving-tests were made in pint fruit-jars of glass, 3 in. in diameter and 6 in. high, the covers being held in place by screw-clamps, and the joints made tight by rubber gaskets. The jars were rotated in the Richards revolving apparatus[10] shown in Figs. 1 and 2. This machine consists of a horizontal shaft with pulley, having at either end a cylindrical box holding 7 jars. Each box has a wooden bottom, 12 staves connected by a stout wire, and sides of 2.5-lb. lead. A jar is held loosely in a horizontal position by three sheet-steel springs, *a*, screwed to the wooden bottom. A circular felt pad, *b*, tacked down between the three springs, protects the bottle against jarring. The barrel makes 4.3 rev. per minute.

[10] *Ore Dressing*, by R. H. Richards, vol. ii., p. 1157.

The Solvents.

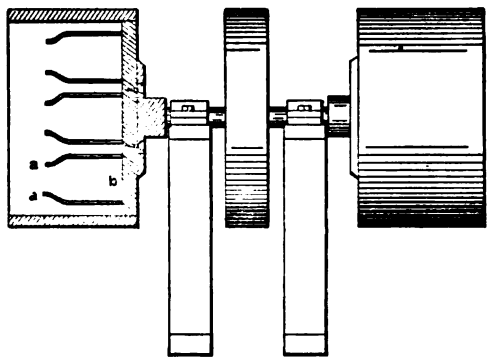
In order to obtain a satisfactory extraction of gold by chlorination or bromination, it is essential to have an excess of reagent over that taken up by the ore. In working on a large scale, it has been found[11] that the consumption of chemicals per ton of ore is the larger, the smaller the charge. Bringing this down to a laboratory-scale, the proportions necessary will be much greater than those found in practice. Considering further the manner of preparing the finely-divided gold or gold-silver, which was coarser than that found in ores subjected to leaching, the excess of solvent in the present case has to be larger than that required by a natural ore in a laboratory-test.

Fig. 1.



END ELEVATION.

Fig. 2.



SIDE ELEVATION AND PART SECTION.
a, wire springs for holding the bottle.
b, place for protective felt pad.

Richards Revolving Amalgamator.

In comparing chlorine and bromine as solvents, it will be necessary to have equivalent quantities. Guided by experiences in regular laboratory practice,[12] the quantities of bromine settled upon, using 150 c.c. of water and 5 A. T. of ore, were 3.0, 2.0, 1.5, 1.0, 0.8, 0.7 and 0.5 c.c. According to Roozeboom,[13] a saturated solution of bromine in water contains at 0° C., 4.05 per cent. of bromine; at 3°, 3.8 per cent.; and at 10°, 3.33 per cent. The solubility decreases slowly with the temperature.

[11] Godshall, *Engineering and Mining Journal*, Jan. 6, 1894, p. 6.

[12] *Notes on Assaying*, by R. W. Lodge. Wiley & Sons, New York, p. 248 (1904).

[13] *Dictionary of Chemical Solubilities*, by Comey. Macmillan & Co., London (1896), p. 62.

Dancer[14] gives for 15° C., 3.226 per cent. of bromine, and for 30°, 3.126 per cent. With 1 c.c. of bromine weighing 2.99 g., the equivalent of chlorine will be 1.32 g., according to the ratio—Br : Cl = 79.96 : 35.45 = 2.99 : x. The chlorine in the experiments was obtained by the action of sulphuric acid upon bleaching-powder. The available chlorine in this was determined[15] by adding to a semi-solution of a weighed quantity of bleaching-powder an excess of a standardized solution of arsenious oxide, titrating back with a standard solution of iodine and calculating the available chlorine from the oxidized arsenious oxide. The bleaching-powder contained 20.78 per cent. of available chlorine, hence the weight of the desired amount of chlorine, in grams, has to be multiplied by 4.81 in order to obtain the necessary quantity of bleaching-powder. Multiplying by 5 gives a slight excess. For each gram of bleaching-powder used, 1.5 g. of concentrated sulphuric acid (sp. gr., 1.84) were added. The solubility of chlorine in water is limited. According to Gay-Lussac :[16]

TABLE I.—*Solubility of Chlorine in Water.*

1 c.c. = 1 g. Water Dissolves at,[x]	Volume.[y]	Weight.[z]
Degrees Centigrade.	C.c.	Gram.
0	1.43	0.0045
8	3.04	0.0096
10	3.00	0.0095
17	2.37	0.0075
100	0.15	0.0002

[x] Only part of temperatures given in the original are reproduced here.
[y] Not corrected to 0° C. and 760 mm.
[z] C.c. of Cl. × 0.00317344 = grams Cl.

MODE OF OPERATING.

In making up a charge, the gold- or alloy-powder was well mixed with the 40-mesh quartzite and charged into the flask. In brominating, 150 c.c. of water was added and then the required cubic centimeters of bromine dropped from a graduated pipette keeping the tip under water. In chlorinating, the

[14] Comey, *op. cit.*, p. 62.
[15] *Quantitative Chemical Analysis*, by H. P. Talbot. Macmillan, New York (1899), p. 111.
[16] Comey, *op. cit.*, p. 105.

bleaching-powder was mixed with the ore, the mixture filled into the flask, 150 c.c. of water added, and lastly the sulphuric acid. Although there is some danger of loss of chlorine in this method of charging, an account of the gas being sometimes evolved before the lid can be secured down firmly, it is to be preferred to charging the bleaching-powder in bulk and then covering it with the ore, as the powder is not so liable to form lumps coated with calcium sulphate, which prevents further action of the sulphuric acid and thus gives less chlorine than called for by the test. The time of rotation was uniformly 5.5 hr., the round box making 4.3 rev. per minute. When the time had elapsed, the content of a jar was transferred to a 7-in. filter and washed for from 2 to 3 hr. with from 800 to 1,000 c.c. of water, ferrous sulphate serving as reagent to test for gold; the residue and filter were placed on a roasting-dish at the mouth of a muffle, dried, the filter burned, the whole passed through a sieve to break up all lumps, then charged with the necessary fluxes into a Battersea crucible, mark K, and fused. The charge, comprising ore, 5 A. T.; sodium bicarbonate, 150 g.; litharge, 300 g.; argols, 10 g.; and the salt cover; gave in 1.5 hr. a lead button weighing about 125 g., which had to be reduced by scorification to about 25 g. before the cupellation. This was done in a 3.5-in. scorifier necessitating one pouring-off of slag. The alloy-buttons were parted in the usual way after adding the required quantity of silver, and fusing before the blow-pipe. The conditions of assay were kept as uniform as possible, in order that any losses by slagging, volatilization and cupel-absorption might be the same, and the results correct in relation to one another.

Results.

The results obtained are given in Table II. for chlorination, and Table III. for bromination, and graphically represented in Figs. 3 and 4 respectively. The tables require no further explanation. In Fig. 3 the constant is the quantity of active reagent, the variables are the percentage of extraction drawn as ordinate and the ratios of gold and silver drawn as abscissa. In Fig. 4, which was plotted to bring out in a more striking way the dissolving-effects of chlorine and bromine, the variables are quantity of reagent drawn as ordinate and the percentage of extraction drawn as abscissa, while the composition of

TABLE II.—*Chlorination of Gold and Gold-Silver with Varying Quantities of Reagent.*

Ore, 5 A. T.; water, 150 c.c.; time, 5.5 hr.

Bleaching Powder.	Sulphuric Acid.	Chlorine (Calculated).	Precious Metal.	Gold in Tailings.	Gold Extraction.	Precious Metal.	Gold in Tailings.	Gold Extraction.	Precious Metal.	Gold in Tailings.	Gold Extraction.	Precious Metal.	Gold in Tailings.	Gold Extraction.
Grams.	Grams.	Grams.	Mg.	Mg.	Per Cent.	Mg.	Mg.	Per Cent.	Mg.	Mg.	Per Cent.	Mg.	Mg.	Per Cent.
14	21	2.8	Gold 10 : Silver 6.	0.04	99.6	Gold 9 : Silver 1.	0.18	98.2	Gold 8 : Silver 2.	0.30	96.3	Gold 7 : Silver 3.	4.18	40.3
10.5	15	2.1		0.14	98.6		0.30	96.7		4.98	37.8		4.40	37.1
7	10.5	1.4		0.75	92.5		0.90	90.0		5.90	26.5		5.10	27.1
6.5	9.75	1.3		5.48	45.2		5.37	40.3		6.23	22.1		6.08	13.2
6	9	1.2		7.44	25.6		6.84	24.0		6.28	21.5		6.15	12.1
5	7.5	1		7.70	23.0		7.09	21.2		6.42	19.8		6.36	9.1
3.5	5.25	0.7		8.06	19.4		7.48	16.9		6.85	14.4		n. d.	.....
2.5	3.75	0.5		8.26	17.4		7.59	15.7		7.12	11.0		n. d.	.....
1	2	0.2		8.56	14.4		7.99	11.2		7.48	6.5		n. d.	.....

TABLE III.—*Bromination of Gold and Gold-Silver with Varying Quantities of Reagent.*

Ore, 5 A. T.; water, 150 c.c.; time, 5.5 hr.

Bromine.	Bromine.	Precious Metal.	Gold in Tailings.	Gold Extraction.	Precious Metal.	Gold in Tailings.	Gold Extraction.	Precious Metal.	Gold in Tailings.	Gold Extraction.	Precious Metal.	Gold in Tailings.	Gold Extraction.
C.c.	Grams.	Mg.	Mg.	Per Cent.	Mg.	Mg.	Per Cent.	Mg.	Mg.	Per Cent.	Mg.	Mg.	Per Cent.
3	8.97	Gold 10: Silver 6.	0.08	99.2	Gold 8 : Silver 2.	0.18	97.8	Gold 7 : Silver 3.	0.16	97.7	Gold 6 : Silver 4.	4.56	24.0
2	5.98		0.18	98.2		0.20	97.5		1.48	78.9		4.92	18.0
1.5	4.48		0.34	96.6		0.24	97.0		3.00	57.1		5.18	13.7
1	2.99		0.68	93.2		0.58	92.8		3.70	47.2		5.16	14.0
0.8	2.60		0.78	92.2		1.88	77.6		4.40	37.1		5.16	14.0
0.7	2.10		6.58	34.2		6.02	24.8		5.95	15.0		n. d.	.....
0.5	1.5		7.62	23.8		6.56	18.0		6.46	7.7		n. d.	.....

TABLE IV.—*Observations Made During Chlorination-Test.*

Chlorine in 150 c.c. of Water.	Extraction of Gold.	Color of Solution.	Odor of Solution.	Pressure of Solution.
Grams.	Per Cent.			
2.8	99.6	Deep yellow.	Very strong.	Strong.
2.1	98.6	Deep yellow.	Strong.	Strong.
1.4	92.5	Deep yellow.	Medium.	Strong.
1.3	45.2	Yellow.	Medium.	Medium.
1.2	25.6	Yellow.	Medium.	Medium.
1.0	23.0	Pale yellow.	Slight.	Weak.
0.7	19.4	Pale yellow.	None.	None.
0.5	17.4	Almost colorless.	None.	None.
0.2	14.4	Colorless.	None.	None.

the alloy is the constant. In analyzing the results of chlorination in Fig. 3, it is seen that with pure gold the solutions containing 2.8, 2.1 and 1.4 g. of chlorine give good extractions, i.e., 99.6, 98.6 and 92.5 per cent.; with 1.3 g. chlorine there is a sudden fall to 45.2 per cent., and with 1.2 g. a second decided lowering of extraction to 25.6 per cent.; from now on the yield

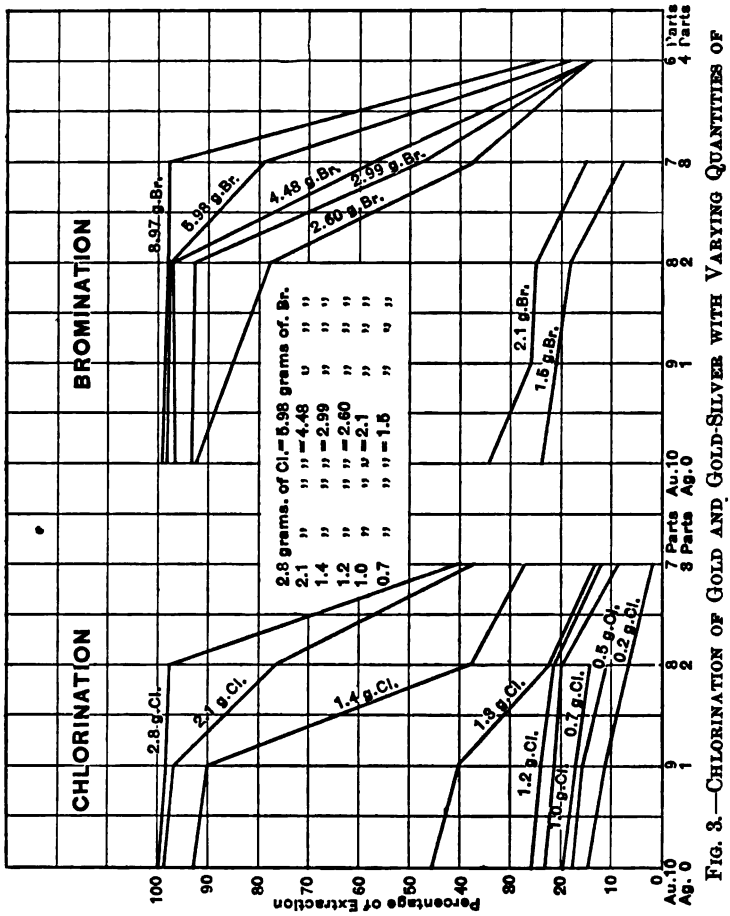


FIG. 3.—CHLORINATION OF GOLD AND GOLD-SILVER WITH VARYING QUANTITIES OF REAGENT.

decreases with the diminishing quantity of chlorine present in more regular way.

Observations made with the flasks during the tests are recorded in Table IV.

Gay-Lussac (see Table I.) has shown that under atmospheric pressure and a temperature of 17° C. 1 c.c. of water can dissolve only 0.0075 g. of chlorine. In order to obtain the super-

saturation required for a good extraction, it is necessary to have a "strong" pressure in the revolving-jar. When the pressure falls to "medium" there is a decided diminution in the yield

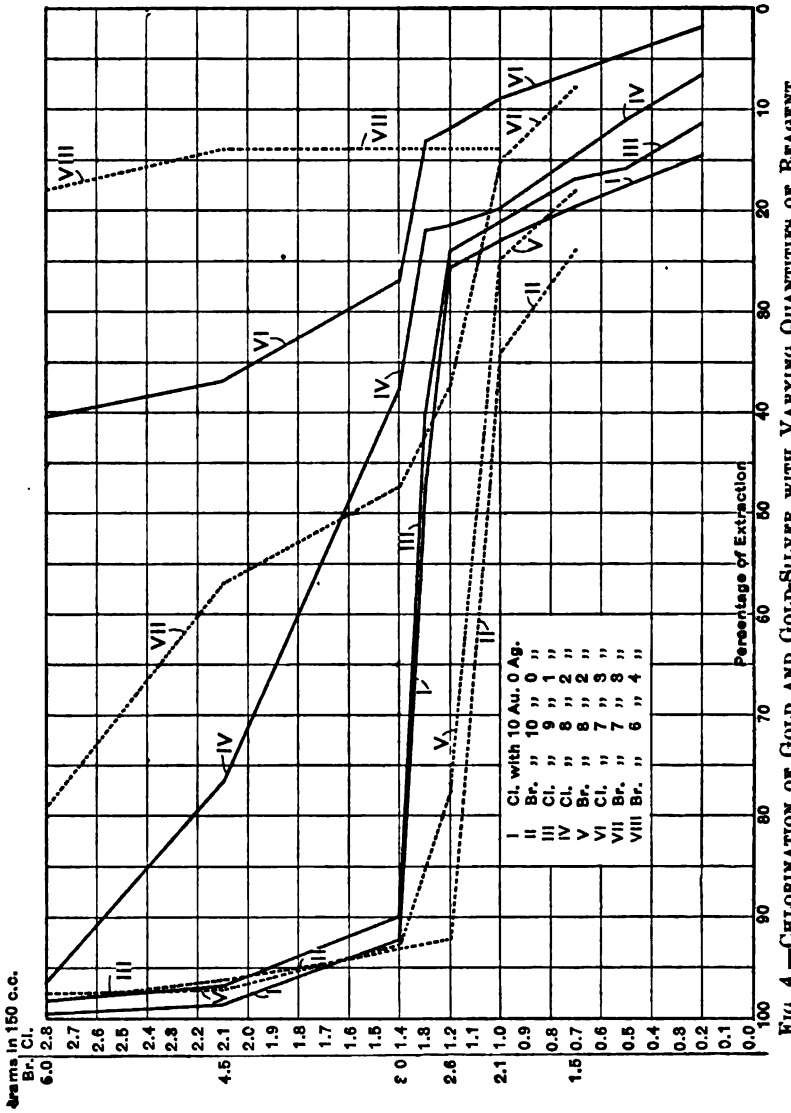


FIG. 4.—CHLORINATION OF GOLD AND GOLD-SILVER WITH VARYING QUANTITIES OF REAGENT.

of gold. This agrees with the experiences made in practice where it has been found that pressure of free chlorine is absolutely necessary for a satisfactory extraction, i.e., one exceeding 90 per cent. As the solution of chlorine in water, according to

the law of Henry, increases with the pressure, one might be led to believe that very high pressures of free chlorine would increase the extraction of the gold, but this has been found not to be the case, as, e.g., at Deloro, Canada,[17] no better results were obtained with 10- than with 60-lb. pressure. The essential point was to have, at first, sufficient pressure of chlorine (10 to 15 lb.) that at the end of the operation (after 2 hr.) there might be some left (3 lb. and less) to maintain the supersaturation of the solvent. The experimental data and results from large-scale work show that a satisfactory extraction of gold in barrel-chlorination can be obtained only by having supersaturated solution of chlorine in water; this requires a certain pressure, varying with the character of the ore, and has to be determined for every case.

In chlorinating the alloy of 9 parts gold with 1 part silver, the extraction with 2.8 g. of chlorine falls 1.4 per cent.; with 2.1 g., 2.1; with 1.4 g., 2.5; and with 1.3 g., 4.9 per cent.; the decrease in yield growing with the diminishing grams of chlorine. With smaller quantities of chlorine, from 1.2 to 0.2 g., there is no such regularity.

With the alloy of gold 8 parts and silver 2 parts, the supersaturated solution with 2.8 g. of chlorine shows only a small decrease of 0.7 per cent. in the gold dissolved. The solution with 2.1 g. of chlorine has its dissolving power diminished by 20.1 per cent., that with 1.4 g. of chlorine by 52.2 per cent.; that with 1.3 g. of chlorine falls off less, viz., 18.2 per cent.; it loses its previously isolated position and unites with the extremely low extractions of solutions with from 1.2 to 0.2 grams.

With the alloy of gold 7 parts and silver 3 parts, the solution with 2.8 g. of chlorine succumbs to the influence of silver, dissolving only 40.3 per cent. of the gold.

The data show that a supersaturated solution of chlorine in water acts more strongly than one that is merely saturated, that a supersaturated solution can extract a satisfactory percentage of gold from a gold-silver alloy containing as much as 20 per cent. of silver, and that with the decrease of chlorine below a certain amount (2.1 g.) and the increase of silver above 10 per cent., the extraction of gold falls off quickly.

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[17] *Engineering and Mining Journal*, Oct. 29, 1887, p. 308.

The results of bromination in Fig. 3 show that bromine also is an efficient solvent for gold, giving extractions of 99.8, 98.2, 96.6, 93.2 and 92.2 per cent. with solutions of 8.97, 5.98, 4.48, 2.99 and 2.60 g. of bromine in 150 c.c. of water; when the bromine present sinks to 2.1 g. the yield in gold falls quickly to 34.2 per cent., and then diminishes more gradually.

Observations made with the flasks during the tests are recorded in Table V.

TABLE V.—*Observations Made During Bromination-Test.*

Bromine in 150 c.c. of Water.	Extraction of Gold.	Color of Solution.	Odor of Solution.	Pressure of Solution.
Grams.	Per Cent.			
8.97	99.2	Dark-red.	Strong.	None.
5.98	98.2	Dark-red.	Strong.	None.
4.48	96.6	Dark-red.	Strong.	None.
2.99	93.2	Dark-red.	Slight.	None.
2.6	92.2	Dark-red.	None.	None.
2.1	34.2	Light-red.	None.	None.
1.5	23.8	Pale-red to yellow.	None.	None.

The experiments of Dancer[18] show that at 15° C. (ordinary temperature), saturated bromine water contains 3.22 per cent. of bromine; a saturated solution of 150 c.c. of water contains 4.839 g. of bromine. The tests show that a solution with as little as 2.6 g. of bromine dissolves more than 90 per cent. of the gold when silver is absent.

The presence of 20 per cent. of silver in the gold does not make more than 1 per cent. difference in the extraction in the first four tests with 8.97, 5.98, 4.48 and 2.99 g. of bromine; the fall begins with 2.6 g. of bromine which dissolve 77.6 per cent. of gold, while in the absence of silver the same amount of bromine gives an extraction of 92.2 per cent.

With an alloy of 7 parts gold and 3 parts silver, only the solution with 8.97 g. of bromine holds its own, giving a yield of 97.7 per cent.; the other solutions show diminishing extractions corresponding with the smaller amounts of bromine present.

With an alloy of 6 parts of gold and 4 parts of silver, the stronger solutions are similar to the weaker ones in their low percentages of extraction.

[18] Comey, *op. cit.*, p. 62.

The data show that 150 c.c. of water containing only 1.93 g. of bromine give a satisfactory extraction (92.8 per cent.) of gold from an alloy of gold with as much as 20 per cent. of silver, and that the extraction-figure is raised only 4.7 per cent. by doubling the amount of bromine.

The comparison of the dissolving-effects of chlorine and bromine upon pure gold and upon gold alloyed with different amounts of silver can be seen in Fig. 3, but is more clearly illustrated in Fig. 4.

Comparing curve I (chlorine and pure gold) with curve II (bromine and pure gold), it is seen that with concentrated solutions (2.8 g. of Cl equivalent to 6.0 g. of Br), chlorine is a slightly stronger solvent than bromine; as the solutions become less concentrated the dissolving power of chlorine falls more quickly than that of bromine, until at 1.58 g. chlorine (equivalent to 3.4 g. of bromine) the power of the two is the same. With further dilution to 1.20 g. of chlorine, bromine is a much better solvent than chlorine; when the dilution is carried very far, the percentages of extraction by the two solvents come again nearer together, but bromine always remains the more efficient of the two.

Curve III shows that the extraction of gold from an alloy of gold 9 parts and silver 1 part by means of chlorine is but little affected by the presence of 10 per cent. of silver; that this is still less the case with bromine can be inferred from the bromination curves in Fig. 3, where pure gold and gold with 20 per cent. of silver are connected by straight lines.

Coming to the alloy of 8 parts of gold and 2 parts of silver, the harm done by the silver on the extraction with chlorine (curve IV) is marked, while with bromine it has very little effect. While with 2.8 g. of chlorine the extraction of gold is diminished only 2.1 per cent., it falls quickly and regularly with dilution of the solvent to 22.1 per cent. until 1.3 g. has been reached, and then falls suddenly to an insignificant figure. With bromine, dilution from 6 to 3 g. hardly affects the result, and a further addition of water only very little. Thus bromine again is the better solvent in the presence of silver.

Curves VI and VII, representing the treatment of an alloy of 7 parts of gold and 3 parts of silver, with chlorine and bromine, show how seriously the extraction of gold is affected by

the presence of such large proportions of silver, but the extractions of bromine are always better than those of the equivalent quantities of chlorine.

Curve VIII, representing the alloy of gold 6, and silver 4, finally shows that even bromine loses its dissolving power when it meets such large proportions of silver.

The curves bring out the additional interesting fact that changes in the degree of concentration of chlorine, as well as of bromine, do not make a very decided difference in the percentage of extraction with pure gold or gold with 10 per cent. of silver, as long as with 150 c.c. of water, 1.4 g. of chlorine, equivalent to 3 g. of bromine, is not overstepped as the lower limit. With gold containing more than 10 per cent. of silver, a slight decrease in the concentration makes a very decided difference in the extraction until 1.2 g. is reached with chlorine, and 2.1 g. with bromine. If the solutions are further diluted, the effect of an increase in silver is not so very marked.

DISCUSSIONS.

Origin of Pebble-Covered Plains in Desert Regions.

A Discussion of the Paper by Mr. William P. Blake (see *Trans.*, xxxiv., 161).

J. COLLETT MOULDEN, Cockle Creek, New South Wales, Australia (communication to the Secretary*):—The pebble-covered plains, which extend over many hundred square miles in the northern arid regions of South Australia and in Central Australia, present a parallel to the similar areas in the Great Colorado of the West, as noted by Mr. W. P. Blake; and his description of the one case, as given in his interesting paper, serves equally well for the other.

The transcontinental railroad, which was originally projected to cross Australia from South to North, starts at Adelaide, and extends, at present, to Oodnadatta, 688 miles distant. The line between Hergott Springs (421 miles from Adelaide) and Oodnadatta traverses mile after mile of pebble-strewn country, which is only occasionally broken by a dry water-course, or "gum-creek," presenting a wearisome outlook to the eye of the traveler. These plains continue further north than Oodnadatta and their total area must be enormous.

The pebbles, in many places, form an almost perfect mosaic, or pavement, with scarcely any interstitial space; and the broad areas covered by them, glittering in the bright sunlight, when viewed from a distance, give the effect of sheets of water.

An examination of the exposed surface of each pebble shows in perfect manner the peculiar greasy polish which results from its attrition by wind-swept sand. The black and dark-brown colors, which Mr. Blake notes in his paper, predominate also here, although the pebbles are composed of widely different materials, the most common being quartz, ironstone, silicified sandstone, chalcedony and agate. The rounded agates often form very attractive specimens, the polishing by the wind having been so perfect as to approach the work of a lapidary. Many, but not all, of the stones are waterworn.

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* Received August 8, 1904.

The pebbles rest upon a sandy soil, through which other and similar ones are distributed; and the origin of the "tesselated pavement" structure which they now present is, without doubt, due to the removal of the soil by wind, which has caused the heavier stones to sink vertically until they become so closely set as to protect the underlying stratum and stop further denudation. The pebbles, composing the upper surface of these stony plains, are commonly called "gibbers," and were of economic importance during the construction of the transcontinental railroad, inasmuch as they formed a very excellent and cheap ballast when merely raked and placed upon the track.

The well-known "obsidian bombs," "obsidian buttons," or "Australites," as they have been variously termed, occur on these "gibber-plains," and latterly their origin has been ascribed to meteoric fall,—a theory supported by much argument. From available palæontological evidence the areas herein referred to appear to have had a fairly abundant rainfall until the later Tertiary times, when, possibly, the waterworn pebbles were formed. At the present time, the rainfall is very small and capricious, with the result that sub-ærial denudation is almost wholly confined to æolian influence. I have observed similar pebble-strewn plains of lesser extent, in other dry and desert places in Australia, more particularly in the districts bordering the Great Victoria desert in Western Australia; and it is clear that the origin of these plains also cannot be otherwise than that given by Mr. Blake for those in the desert regions of the United States,—in fact, they can be watched in process of formation to-day.

Concrete in Mining and Metallurgical Engineering.

A Discussion of the Paper by Henry W. Edwards, p. 60.

EDWIN H. MESSITER, New York City (communication to the Secretary*):—Under the heading "Flues," Mr. Edwards refers to the Bee-hive construction, a cross-section of which is shown in Fig. 4 of his paper. A flue similar to this was designed by me about six years ago,[1] in which the walls though much thinner than those described by Mr. Edwards gave entire satisfaction. These walls, from 2.25 in. thick throughout in the smaller flues, to 3.25 in. in the larger, were built by plastering the cement mortar on expanded-metal lath without the use of any forms, or cribs, whatever, at a cost of labor generally less then $1 per sq. yd. of wall. Of course, where plasterers cannot be obtained on reasonable terms, the cement can be molded between wooden forms, though it is difficult to see how it can be done with an interior core only as stated by Mr. Edwards.

In regard to the effect of sulphur dioxide and furnace-gases on the cement, I have found that, in certain cases, this is a matter which must be given very careful attention. Where there is sufficient heat to prevent the existence of condensed moisture inside of the flue, there is apparently no action whatever on the cement, but if the concrete is wet, it is rapidly affected by these gases. At points near the furnaces there is generally sufficient heat not only to prevent internal condensation of the aqueous vapor always present in the gases, but also to evaporate water from rain or snow falling on the outside of the flue. Further along, a point is reached where rain-water will percolate through minute cracks caused by expansion and contraction, and reach the interior, even though internal condensation does not occur there in dry weather. From this point to the end of the flue the roof must be coated on the outside with asphalt paint or other impervious material. In very long flues a point may be reached where moisture will condense on the inside of the walls in cold weather. From this point to

* Received March 14, 1904.

[1] *Engineering News*, Nov. 30, 1899, p. 356, and U. S. Patent No. 665,250, Jan. 1, 1901.

the end of the flue it is essential to protect the interior with an acid-resisting paint, of which two or more coats will be necessary. For the first coat, a material containing little or no linseed oil is best, as I am informed that the lime in the cement attacks the oil. For this purpose I have used ebonite varnish, and for the succeeding coats, durable metal-coating. The first coat will require about 1 gal. of material for each 100 sq. ft. of surface.

In one of the earliest long flues built of cement in this country, a small part near the chimney was damaged as a result of failure to apply the protective coating, the necessity for it not having been recognized at the time of its construction. It may be said, in passing, that other long brick-flues built prior to that time were just as badly attacked at points remote from the furnaces. In order to reduce the amount of flue subject to condensation, the plastered flues have been built with double lath having an intervening air-space in the middle of the wall.

In building thin walls of cement, such as flue-walls, it is particularly important to prevent them from drying before the cement has combined with all the water it needs. For this reason the work should be sprinkled freely until the cement is fully set. Much work of this class has been ruined, through ignorance, by fires built near the walls in cold weather, which caused the mortar to shell off in a short time.

The great saving in cost of construction, which the concrete-steel flue makes possible, will doubtless cause it to supersede other types to even a greater extent than it has already done. If properly designed this type of construction reduces the cost of flues by about one-half. Moreover, the concrete-steel flue is a tight flue as compared with one built of brick. There is a serious leakage through the walls of the brick flues which is not easily observed in flues under suction as most flues are, but when a brick flue is under pressure from a fan, the leakage is surprisingly apparent. In flues operating by chimney-draft the entrance of cold air must cause a considerable loss in the efficiency of the chimney, a disadvantage which would largely be obviated by the use of the concrete-steel flue.

FRANCIS T. HAVARD, Silberhütte, Anhalt, Germany (communication to the Secretary*):—In discussion of Mr. Edwards's

* Received July 8, 1904.

interesting and valuable paper, I beg to submit the following notes concerning the advantages and disadvantages of the concrete flues and stacks at the plant of the Anhaltische Blei-und Silber-werke. The flues and smaller stacks at the works were constructed of concrete consisting generally of one part of cement to seven parts of sand and jig-tailings, but in the case of the under-mentioned metal-concrete slabs, of one part of cement to four parts of sand and tailings. The cost of constructing the concrete flue approximated 5 marks per sq. m. of area (equivalent to $0.11 per sq. ft.).

Effect of Heat.—A temperature above 100° C. caused the concrete to crack destructively. Neutral furnace-gases at 120° C., passing through an independent concrete flue and stack, caused so much damage by the formation of cracks that, after two years of use, the stack, constructed of pipes 4 in. thick, required thorough repairing and auxiliary ties for every foot of height.

Effect of Flue-Gases and Moisture.—The sides of the main flue, made of blocks of 6-in. hollow wall-sections, 100 cm. by 50 cm. in area, were covered with 2-in. or 1-in. slabs of metal-concrete. In cases where the flue was protected on the outside by a wooden or tiled roof, and inside by an acid-proof paint, consisting of water-glass and asbestos, the concrete has not been appreciably affected. In another case, where the protective cover, both inside and outside, was of asphalt only, the concrete was badly corroded and cracked at the end of three years. In a third case, in which the concrete was unprotected from both atmospheric influence on the outside, and furnace-gases on the inside, the flue was quite destroyed at the end of three years. That portion of the protected concrete flue, near the main stack, which came in contact with dry cold gases only, was not affected at all.

Gases alone, such as sulphur dioxide, sulphur trioxide, and others, do not affect concrete; neither is the usual quantity of moisture in furnace-gases sufficient to damage concrete; but should moisture penetrate from the outside of the flue, and, meeting gaseous SO_2 or SO_3, form hydrous acids, then the concrete will be corroded.

Effect of the Atmosphere Alone.—For outside construction-work, foundations and other structures not exposed to heat, moist acid-gases and chemicals, the concrete has maintained its reputation for cheapness and durability.

Effect of Crystallization of Contained Salts.—In chemical works, floors constructed of concrete are sometimes unsatisfactory, for the reason that soluble salts, noticeably zinc sulphate, will penetrate into the floor and, by crystallizing in narrow confines, cause the concrete to crack and the floor to rise in places.

MR. HENRY W. EDWARDS, Grand Junction, Colo. (communication to the Secretary*):—I beg to present the following notes in reply to Mr. Havard's discussion:

Effect of Heat.—It should be remembered that the coefficient of expansion of concrete is nearly that of iron; therefore in constructing flues the design must take account of this important detail. The cracking of flues, mentioned by Mr. Havard as having occurred at a temperature of 100° C. and above, were possibly caused by neglecting to appreciate the extent of the expansion and contraction of the concrete.

Effect of Moisture.—The moisture present in the atmosphere, as well as that derived from roasting moist ores, is exceedingly injurious to flues of any material, and those made of concrete are no more susceptible than of brick. My experience with flues of the Monier construction, having walls from 1.5 to 2.5 in. thick plastered upon a net-work of iron rods, was unsatisfactory from the standpoint of durability.

Fuel and Mineral Briquetting.

A Discussion of the Paper by Robert Schorr, p. 82.

E. T. DUMBLE, Houston, Texas (communication to the Secretary†):—In addition to the list of publications mentioned by Mr. Schorr and those by Prof. Hofman, I call attention to the following references:—

Studi sulle Lignite, by Capacci. Turin (1890). (An excellent work.)

Report of Brown Coal Industry in Germany. J. Cosmo Newberry. Department of Mines, Victoria (1892).

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* Received October 25, 1904. † Recived Aug. 27, 1904.

Second Report of the Royal Commission on Coal Supply. Vol. II. London (1904).

Peat Fuel: Its Manufacture and Use. Report of the Bureau of Mines of Ontario, 1903.

Mr. Schorr says: "Brown coal and peat are briquetted as a rule without binder," but this statement is entirely true only so far as raw peat is concerned, and it is but partly true of brown coal.

With regard to raw peat, briquetting without bond is carried on successfully both in Europe and America, and the development of the industry in Canada is fully described in the Bureau of Mines report for 1903, mentioned above. While there is still room for improvement in the efficiency of some of the arrangements for drying the peat previous to briquetting, a very satisfactory fuel is made at a price which enables it to compete with coal.

During the late strike in the anthracite regions of Pennsylvania, the manufacture of peat-briquettes received a considerable impetus, and possibly the attention directed to it on this account may result in a considerable increase in the output. In Canada, the peat when it is first excavated from the ground contains more than 80 per cent. of moisture, which quantity is reduced, first by drying in air and then in artificial dryers until in the finished briquette there remains only 15 or 16 per cent. of water. The briquettes are cylindrical in shape, 2 in. long and 2 in. in diameter, contain from 1.55 to 10 per cent. of ash and were said to have a calorific value two-thirds that of anthracite coal. The estimated cost of excavating, drying and pressing peat is $1.01 per short ton, while the gross cost, including land, depreciation, interest and royalty (25c.) amounts to $1.80 per ton.

The earthy brown coal of the Rhine provinces and in the vicinity of Halle am Saale is the only variety of brown coal, so far as I know, which can be briquetted without bond. This variety of brown coal is very different from common brown coal and its derivatives, pitch-coal and glance-coal, and it is even claimed by some that earthy brown coal is not a true coal at all. It frequently contains retinite, amber and the pyropissite or "schweelkohle" which is the basis of the paraffine industry of Germany. Schweelkohle does not occur in other

brown coals. In structure it is usually friable and earthy, although at times it is more compact and resembles common brown coal. In color it varies from yellow to brown or brownish black. Earthy brown coal when first it is mined contains from 30 to 60 per cent. of moisture, which is considerably reduced by drying in the process of briquetting, but the finished briquettes still carry from 14 to 18 per cent. of water. Consequently, after earthy brown coal has been briquetted, it is practically of the same composition as the common brown coal of Bohemia and Texas taken direct from the mine, but it is firm and clean, it will not slack on exposure to the air and it burns with little smoke, and, for these reasons, it is a very desirable fuel.

In the briquetting-works of Germany, of which there are 286, more than 44,000,000 tons of earthy brown coal are annually consumed.

So far as I know there is no earthy brown coal in the United States.

Common brown coal, pitch-coal and glance-coal, which constitute by far the greater bulk of all the brown-coal deposits, cannot be briquetted without bond under ordinary conditions; and numerous failures are on record of attempts to make serviceable briquettes of them even with the best of binders.

In connection with briquetting brown coal with a bond, Mr. Schorr refers to the lignite-briquetting plant built at Rockdale, Tex.

Under my direction the brown coal from Rockdale was experimented with in Europe in 1892, and a good firm fuel made from it by briquetting with a binder of hard pitch without other admixture. Therefore, there is every reason to expect that satisfactory briquetting can also be done in the United States by proper treatment. Later, the plant at Rockdale was taken up by San Antonio people, and a firm of mechanical engineers, of considerable experience in coal, was employed to investigate the subject of briquetting and to construct a plant. In 1896, after an examination of the various machines and appliances used in Europe, a plant was designed and erected which, it was believed, would meet the requirements; but so far as I can learn it never passed beyond the experimental stage. The first trouble encountered was to get a sufficient quantity of pitch at

a suitable price; but this product was finally obtained from the iron blast-furnaces in Scotland. Then the quantity of moisture in the lignite could not be reduced to the exact point necessary for a good cohesion. In October, 1896, the plant burned down, but it was later rebuilt.

My information is that a fuel could not be made of sufficient firmness to be satisfactory. Whether this was the fault of the press or of the treatment, I am not prepared to say, but the work was considered a failure. Such a failure as this, backed as it was by considerable money, has made capital wary of a second investment in this attractive field. Nevertheless, I am convinced that sooner or later the process of briquetting Texas brown coals with a binder of pitch or asphaltum will be successfully operated and that it will provide a satisfactory fuel for general use.

The Equipment of a Laboratory for Metallurgical Chemistry in a Technical School.

Discussion of a Paper by Mr. C. H. White, p. 117.

ARTHUR JARMAN, Sydney, New South Wales, Australia (communication to the Secretary*):—All designs for modern metallurgical and chemical laboratories should provide each student's desk with a hood or cupboard for the removal of fumes, but I doubt the advisability of placing air-baths and similar apparatus close to the hot plate. It seems to me that unless the draft be exceptionally strong the fumes from ordinary wet assay work will corrode the iron-work of the fittings, in which case the baths would have to be removed to a general desk in order to lighten the work of the laboratory attendant. Since the laboratory described by Mr. White has been in use for two years, I should like to ask if any inconvenience has resulted from this corrosion. If not, it is certainly an important improvement to have the air-baths and similar apparatus placed on the student's desk.

The hoods described by Mr. White are supplied with a good draft by means of a fan, which gives them a great advantage

* Received October 17, 1904.

over those at the metallurgical laboratory at the University, Sydney, in that no doors to the hood are needed. Each hood in the laboratory at Sydney is connected by a glazed earthenware drain-pipe with a vertical flue, 20 ft. high, having a lighted gas-jet at the bottom to assist the draft. Although the draft is good, fumes of sulphuric acid cannot be drawn off unless the doors of the hood are lowered more than half-way. In my opinion, a fan should be used for the purpose of creating the draft, and glazed drain-tile should invariably be used for the flues because no other material possesses the same advantages.

I know of a recent instance in which a laboratory containing about 20 desks has been fitted with flues of sheet-iron, tarred inside and kalsomined on the outside. How long these will last is a question, but from personal experience with flues of this character I should say that they will crumble to pieces within three years.

The floors of the hoods of the University laboratory at Sydney are covered with 6- by 6-in. white glazed tiles, $\frac{3}{8}$ in. thick, set in asphalt directly on the top of the desk. Should the desk-top become warped by the heat from the burner and hot plate, the tiles may be disturbed, although in seven hoods after four years of service only two tiles had become sufficiently displaced to allow acid to make its way through the joints. A foundation that will not buckle when heated is a needed improvement, and for this purpose cement would answer, the tiles being set in a small quantity of asphalt in order to prevent acids from attacking the joints. Since Mr. White does not specify the method of placing the tiles in the laboratory at Harvard, I should like to ask what construction was finally adopted.

With regard to the means used to furnish exhaust and pressure, each desk in the laboratory of the University at Sydney, accommodating four students, is equipped with a water suction-pump which gives, according to tests with a mercury-gauge, a 680-mm. vacuum or an actual pressure of only 80 mm. Occasionally, better results than these have been obtained, but pumps of this kind have the inherent defect that rust from the water-supply pipes will clog them after a long period of inaction. The use of a blowing-engine to furnish both suction and air under pressure removes this difficulty and also avoids the variation in pressure which inevitably accompanies the use of water-blowers or suction-apparatus.

At Sydney much difficulty was experienced in obtaining a satisfactory lock for the desks. Finally we adopted a circular brass plate which covers the edges of the doors, and is firmly secured by a pad-lock. Other locking-devices can often be removed by the use of an ordinary screw-driver.

Mr. White does not mention the detailed construction of the sinks in the laboratory at Harvard. The usual lining of sheet lead becomes quickly corroded by acid and other liquids, and at Sydney the cost of plumbing repairs became large. In order to reduce this item, Mr. Schofield, Demonstrator in Chemistry at the University at Sydney, introduced the use of a false bottom, over a layer of broken marble. The false bottom was first made of perforated sheet-lead, but as this was rapidly eaten away, it was later replaced by glazed earthenware trays. The use of marble under a false bottom has saved much trouble and has considerably reduced the cost of plumbing-repairs at Sydney.

A Decade in American Blast-Furnace Practice.

A Discussion of the Paper of Mr. F. Louis Grammer, p. 124.

EDWARD A. UEHLING, New York City (communication to the Secretary*):—In adding my mite to the discussion, I wish to touch on a few points which bear emphasizing and perhaps a little further elucidation. Mr. Grammer says that he "found the electric pig-breaker at Duquesne, if used in connection with the iron-chill, less expensive and more satisfactory" than the casting-machine.

The cost of handling the iron over the Duquesne casting-machine is about double the average cost of the other Uehling machines at the several Carnegie plants. It is nearly as expensive to run as the Heyl & Patterson machines, and while Mr. Grammer's statement is, no doubt, true, as applied to that particular machine and breaker, it should be borne in mind that the machine at Duquesne had embodied in it several new features, which were not improvements; also that it was taxed beyond its economic capacity.

The machine at the Lucy furnaces under the management of Mr. Scott handled the output of the two furnaces, and saved

* Received March 23, 1904.

fully 10c. a ton over handling by hand. The machine at the Monongahela furnaces at McKeesport handled the iron for less than 10c. per ton, including all repairs. At Struthers, a single-strand Uehling machine has been taking all the iron (from 300 to 320 tons daily) that the furnace has been making since 1897. An itemized cost-account of all labor, repairs, etc., shows that the difference in the cost in favor of the machine, over what it would cost to handle the iron by hand at current wages, more than covers the cost of the installation every two years.

The cost of handling iron over the Davis casting-machine is somewhere between the other two. At the Bethlehem Steel Co., the record shows a saving of from 2 to 5c. per ton over handling by hand. At another plant where the Davis machine is in use, I am informed that no direct saving is made over handling by hand. Nevertheless, the indirect advantages make the machine a very good investment.

The enormous outputs of the modern furnace have made the casting-machine a necessity, irrespective of the cost of handling the iron; though a large saving over hand-labor can be effected even at smaller plants by selecting the proper type of machine and running it with the required care and intelligence.

Mr. Grammer disposes of the question of flue-dust and gas-washing in a brief paragraph. And so far as any real advance made in the last decade is concerned, either in recovering the former or applying the latter, not much more can be said. Briquetting the flue-dust has not proved commercially successful. Filling it back into the furnace, as fast as produced, is largely practiced with varying and, on the whole, indifferent success. The difficulties arising from the use of the large percentage of fine ores in the mixture are very materially aggravated by filling back the flue-dust. Hence the problem of treating the latter so that it can be utilized without serious difficulty still remains unsolved. Since from 2 to 5 per cent. of the burden (depending on the percentage of Mesabi and other fine ores in the mixtures and also on the size and condition of the furnace) is blown over, the solution of this question is of vast importance.

It has been proposed to mix the dust with sufficient proportion of pulverized slag, and expose the mixture to a tempera-

ture high enough to melt the latter, thus agglutinizing the mixture. I believe that with the proper means, the dust can be agglutinized by hot molten slag direct without the aid of external heat, with greater probability of commercial success.

Far more important, however, than the preparation of the flue-dust, is the proper purification of the gas. In the United States nothing of significance has been done in that direction during the past decade. It is a fact that, with but few exceptions, the gas of the modern blast-furnace carries more dirt into the stoves and under the boilers to-day than was the case a quarter of a century ago.

It is true the dust-catchers have been increased in size and dust-pockets have been multiplied; but the subject has not received the attention that its importance demands. It has been entirely overlooked that, with clean gas, the heating surface of every hot-blast stove could be doubled and the steaming-capacity of every boiler increased at least from 30 to 50 per cent., and that the capital now being invested in additional stoves and more boilers, which the heavy repairs and frequent stoppages, caused by the dirty gas, make necessary, would in most cases be more than sufficient to install an efficient washing-plant. There is no improvement that could be suggested in connection with the modern blast-furnace, that would yield a greater return from the investment than an efficient gas-washing plant, except the blast-furnace gas-engine, and this latter must necessarily be served with clean gas.

In regard to the available power from blast-furnace gas, I am in a position to say that my estimate of 800 h.p. per ton of iron per hour is conservative, providing all the gas is properly utilized. Since writing the article referred to by Mr. Grammer,[1] I have investigated the gases of a number of large furnace-plants, and have in every case found more power in the gas than my estimate called for.

The power available depends principally on the fuel-consumption. In my estimate, I assumed 2,000 lb. of fuel per ton of iron, and in all the plants investigated, the consumption of coke exceeded this quantity.

A good beginning has been made at the Buffalo plant of the

[1] The Blast-Furnace as a Power-Plant, by Edward A. Uehling, *Stevens Institute Indicator*, January, 1903.

Lackawanna Steel Co., and it is incredible that the 1,500,000 to 2,000,000 h.p., now going to waste, should be left unutilized, when such firms as the Allis-Chalmers Co. stand ready to furnish the most approved type of blast-furnace gas-engine of any desired power up to 6,000 h.p. The De la Verne Refrigeration Machine Co., which is building the 1,000- and 2,000-h.p. engines that have been and are now being installed at Buffalo, is, no doubt, ready to contract for others of equal or greater power.

The number of blast-furnace gas-engines now in successful operation abroad, and the rate at which they continue to displace the steam-engines at Continental plants, is proof positive that the experimental stage has been passed, and that it is now only a question of selecting the best from the several types that have been developed.

Referring to furnace dimensions developed in the past decade, I concur fully with Mr. Grammer in his statement:—"Our high furnaces do not reflect great credit on their designers, though in justice it should be said that most furnacemen were not in favor of 100-ft. heights."

From a mechanical standpoint, the modern blast-furnace with its manifold labor-saving appurtenances commands the highest admiration, but from a metallurgical point of view they are monuments of misconception. In the mania for larger and larger units, and greater and still greater outputs, the chemistry of the blast-furnace has been sadly neglected, and the nature of the raw material has not received proper attention. The result is higher fuel-consumption, greater waste in flue-dust, etc., greater irregularity in quality, increased wear and tear, and an aggravation of all the troubles to which blast-furnaces are naturally kin, besides adding new ones.

The design and construction of the Duquesne furnaces, the pioneer high furnaces, was at the time a courageous and laudable undertaking, and the above criticism does not fully apply there. The Mesabi ores had not then entered as a factor in the ore-mixtures, and hence could not be taken into account.

As an experiment, suggested and carried through by the ablest blast-furnace experts, and backed by the strongest company in the United States, the construction of these high furnaces was fully warranted at the time, and if the new ore-finds had been coarse and refractory, instead of very fine and exceed-

ingly easily reducible, the Duquesne furnace-plant would have proved an enormous step in advance.

Because of their physical condition, as well as their chemical composition, the Mesabi ores are the most easily reducible ores in existence. Both the physical and chemical properties call for a decrease, rather than an increase, in the height of furnace, which was recognized as the standard prior to the advent of the Mesabi ore. There are but few ores that require so little time for complete reduction and carburization, and none that cause as much trouble by over-exposure. Except for coarse and very refractory ores, it is a serious mistake to build a furnace more than from 75 to 80 ft. high. The best work, both as to fuel economy and output per unit of cubic capacity, has been done by furnaces less than 75 ft. high.

In designing a blast-furnace, the physical and chemical properties of both ore and fuel must be carefully studied and logically considered in connection with the kind of iron to be produced. Millions of dollars have been worse than wasted in the erection of furnaces of excessive height, which could have been saved if the problem had been properly studied in advance and the logical conclusions followed, instead of blindly copying others with the general idea of "going one better."

Thoughtful managers have recognized the mistake and are beginning to correct the error, and it is safe to predict that, before the end of the present decade, the majority of the high furnaces will be reduced in height to 80 ft. or less.

The Use of High Percentages of Mesabi Iron-Ores in Coke Blast-Furnace Practice.

Discussion of the Paper of W. A. Barrows, Jr., p. 140.

F. E. BACHMAN, Port Henry, N. Y. (communication to the Secretary*):—In discussing Mr. O. O. Laudig's paper, the Action of Blast-Furnace Gases Upon Various Iron-Ores,[1] I took the ground that Mesabi-ore explosions were entirely mechanical; but more observation and later experience has caused me to

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* Received March 7, 1904.

[1] *Trans.*, xxvi., 269.

modify my views. I am still of the opinion that many of the puffs of gas, which at times throw out small quantities of stock, are largely the result of mechanical action, and are caused by the fine ore and deposited carbon filling the voids between the pieces of coke to such an extent that the gas is held back till the pressure is raised to a point which forces it through.

The full-fledged Mesabi-ore explosion seems to come from a point too low in the furnace to be caused by an obstruction of the gas-passage by fine ore and deposited carbon. They are not true explosions for the reason that I have seen stock pouring from explosion-doors during a period of 15 seconds and even longer. If these were gas-explosions, the pressure developed would certainly have burst the furnace before it could have emptied it.

After one of these explosions, or "blows" as they are called in charcoal practice, I have seen coke appear at the tuyeres in three hours from the time it was charged in the furnace, which under ordinary conditions was being emptied once in eight hours. This circumstance would indicate that five-eighths of the material in the furnace had been blown out and that the explosion originated at or near the top of the bosh. An explosion at a 60-ft. charcoal furnace, using three-quarters of Mesabi ore, one-half of it as good physically as Old Range ore, and the other half, equaling the worst of the Mesabi, removed so much material that, after charging more than enough stock in four hours to fill the furnace, it was left out of reach of a 22-ft. rod. At this point the engines were slacked and the furnace was filled rapidly. I think this furnace was practically emptied to the tuyeres. The rate at which the furnace was being driven would empty it every six hours.

Explosions very similar to these are of common occurrence at charcoal furnaces using both Old Range and other ores. I learned of one not long ago which removed every pound of stock to a point 18 in. below the cinder-breast. This result seems impossible, but it cannot be questioned as the brick from the top were removed through the cinder-breast, and no stock except a charge which was on the bell was taken out. To me these observations indicate an evolution of gas, caused by the rapid combustion of fine carbon at the expense of oxygen from

ferric oxide, which has reached a point of sufficient heat in the furnace, without having been deoxidized, to cause a very rapid reduction of the ore, and in consequence an excess of gas. Experiments on the reduction of powdered ore by powdered coke, to which my attention has been called, have indicated that such an evolution is possible.

A uniform quantity of air blown into a furnace and combining with carbon as it is blown, does not cause an explosion. Some other source of oxygen must therefor be sought, and, until data is available concerning the composition of the gas during a deep-seated blow, the cause cannot be positively determined.

That fine ore, *per se*, does not cause explosions is shown by their absence at furnaces using large percentages of magnetic concentrates. For two years I have used a mixture containing from 50 to 66.6 per cent. of magnetic concentrates, all of which passed a 6-mesh sieve, and 90 per cent. of the total quantity of ore charged passed a 0.5-in. mesh, and I have never seen a piece of coke in the flue-dirt, or a puff of brown smoke which was not accompanied by a decided slip.

Regarding the filling of fine ore, my attention was called to the difficulty of properly distributing a very fine ore of uniform size, at a charcoal furnace now using from 92 to 93 per cent. of magnetic concentrates, which would pass a 6-mesh sieve, the remainder of the ore-charge being crushed to pass a 1.5-in. mesh sieve. In charcoal practice, for some reason, from 1,000 to 1,200 lb. is generally accepted as the proper weight for the charcoal charge. At the furnace in question, the charge was 72 bushels (1,300 lb., or about 117 cu. ft.), the ore-charge 2,100 lb. (about 10.5 cu. ft.), the stock-line was 7 ft. 6 in. in diameter (area 45.36 sq. ft.), and the diameter of the bell was 5 ft. As the charcoal dumped from a bell of this size takes the shape of an inverted cone, the area to be covered by the 10.5 cu. ft. of ore was about 55 sq. ft. It is therefore inconceivable that any of it reached the center of the furnace, a conclusion which was confirmed by the working of the furnace. On increasing the volume of the charcoal-charge to 156 cu. ft., filling the ore on top of instead of under the coal, and doubling the quantity of blast, the work of the furnace was gradually improved until the ore-charge was double the weight of the

coal, and was equal to about 17 cu. ft. in volume. This furnace is still working with a very open center, as is indicated by the blast-pressure, which only increased 1.25 lb. to an average of 4.25 lb., being the result of increasing the ore-charge one-third and doubling the quantity of blast. The charcoal-furnace referred to is now using 100 per cent. of 6-mesh magnetic concentrates which is the highest attainable proportion of fine ore in a charge.

This experience led me to make a sectional model on a scale in which 0.5 in. equals 1 ft., of a 70-ft. furnace, having a 17-ft. bosh, 11 ft. 6 in.-diameter of stock-line. The charge used was coke 1, ore 1.6, and stone 0.48 by weight, the coke-charge being proportioned to a 4,800-lb. charge in the furnace. With this size of charge I could get a fairly uniform distribution of coke, but could not get any ore to the center of the furnace. On increasing the coke-charge proportionately to equal 6,000 lb., I obtained a good distribution, with magnetic concentrates which passed a 20-mesh sieve and remained on a 40-mesh sieve, using bells 4 in., 3.75 in., 3.5 in., and 3.375 in. in diameter. With a bell of a diameter smaller than 3.375 in., the ore and the stone would not reach the side or center of the furnace. Using the same sized coke- and ore-charge, but with ore of a size that would pass a 40-mesh sieve, I could not get an even distribution of the ore with a bell of any size.

Any of the bells above referred to, threw the ore to the walls on a 2-in drop, but none of it reached the center of the furnace. With a 3-in. bell a bench of coke was formed against the wall and an inverted cone in the center of the furnace. The throw of the ore was always greater than the throw of the coke; therefore, more ore was near the wall with a 3-in. bell than with a 4-in. bell. With a bell smaller than 3 in., a ridge of coke was formed which sloped to the wall and to the center, the ore striking the coke on the outer side of the ridge again failed to reach the sides, there being a ring of ore with coke inside and outside of it. Sandwiching the charge on the bell and placing the ore and stone on top of the coke, the whole being lowered into the furnace together, improved the distribution with large and small bells, but in all cases a central core of coke was left, much smaller, however, than when the components of the charge were dropped separately; at the same time there was a deficiency of ore against the walls of the furnace.

It is to be noted that a sectional model exaggerates the difficulty of getting the material to the center of the furnace, owing to the area under the bell being almost double its proportional area in a circular section. The experiments referred to, therefore, while not conclusive, indicate the way to work on a full-sized model.

On trying a Killeen distributer with the model, it was found that, when the furnace was kept full, the results were better than those obtained by the ordinary cup and cone; the ore reached the sides, and the center column having no ore was much smaller. If the stock-level was dropped 7.5 in., equaling 15 ft. below the bell in a full-sized furnace, a perfect distribution resulted.

The best distribution obtained resulted from the use of a 4-in. cylinder, with a bell 2 in. in diameter. This arrangement, I think of German origin, was first called to my attention by a German engineer in 1897. It distributed all the stock uniformly, regardless of the level at which the stock was held. There was, however, a tendency to form a central core of coke, although this was much smaller than with any other type of filling-apparatus tried.

EDWARD A. UEHLING, New York City (communication to the Secretary*):—The method of filling a blast-furnace has always been one of the most important elements in its successful management; and Mr. Barrows strikes the key-note of proper filling in his statement:—"The purpose in moving the siliceous-ore, or coarse-ore barrow one pocket each charge is to destroy the continuity of the gas-currents ascending through the fine ore, and to reduce the pressure incident thereto, also to prevent, in a measure, the dust from being carried over into the downcomer."

The stock must be charged into the furnace in such a manner that the current of ascending gas cannot readily plough its way through in a straight line. Homogeneous filling is compatible only with very moderate driving.

From a study of the chemical reactions which take place in a blast-furnace, I came to the conclusion many years ago that stratification would be more conducive to fuel-economy and regularity of working than the more or less thorough com-

* Received March 23, 1904.

mingling of the ore, fuel and flux which was the method then generally in vogue. The theoretical considerations which led me to advocate and to practice the stratified filling (always with very materially improved results) were based on the principal chemical reactions which constitute the process of reduction in the blast-furnace.

The iron-ore Fe_2O_3, for example, can be, and is, reduced according to the following reactions:—

(1) $Fe_2O_3 + 3C = 3CO + 2Fe$.

(2) $Fe_2O_3 + 3CO = 3CO_2 + 2Fe$.

(3) A combination of reactions (1) and (2).

Manifestly, the best possible results, so far as fuel-economy and process of smelting are concerned, is obtained when the maximum amount of the carbon charged is burned to carbon monoxide before the tuyeres, and the ore is reduced by the carbon monoxide thus formed. The heat developed in thus burning the carbon in the hearth produces the necessary temperature in the zone where it is required to melt the iron and the accompanying slag; the carbon monoxide formed, with accompanying nitrogen, in their upward course pre-heating the descending material, thus offering the greatest possible opportunity for the reaction indicated in equation (2), and permitting the gases to escape with a maximum content of carbon dioxide. On the other hand, if the ore is reduced by direct contact with the carbon, that is, according to equations (1) or (3), the hearth is robbed of so much fuel, which is detrimental to the process of smelting, and all the carbon monoxide so formed passing off as such, enriches the escaping gas and is to that degree prejudicial to fuel-economy.

From these considerations it follows that it must be conducive to fuel-economy and improved smelting to reduce the opportunity of direct reduction as much as possible. To accomplish this, I proposed to fill the coke and ore in separate strata, making the layers as heavy as is consistent with existing conditions, and, so far as I am aware, this method of charging a blast-furnace had not previously been proposed, much less systematically practiced.

It so happens that stratified filling was first tried and its virtues proven more than 20 years ago in the same furnace, then known as Douglas furnace No. 1, in which Mr. Barrows

proved the method of filling described in his paper. I had been connected with the Douglas furnaces in 1880, but at the time I was chemist for the Bethlehem Iron Co. Mr. Briorty, furnace-superintendent at Bethlehem, did not feel inclined to try any experiment, but Mr. George Kelly, manager of the Douglas furnaces, to whom I explained the method and reasons for likelihood for success, took it up at once, and I can do no better than quote from his letters in which he reports the results obtained:

"DOUGLAS FURNACES,
"PIERCE, KELLY & Co.,

"SHARPSVILLE, MERCER Co., PA., September 23, 1884.

"E. A. UEHLING, ESQ., Bethlehem, Pa.

"*Dear Sir*:—You no doubt think it about time that you should hear some sort of a report of the results of trial of your method of filling. Two weeks ago yesterday we began filling 12 barrows of coke at a time, and 12 barrows of ore with 6 barrows of lime, which gives us an average depth of say 12 to 15 inches. We have kept this up steadily since then with very gratifying results. The first week the furnace was on cinder-mixture and the second week on all-ore mixtures, and in each case there has been a most marked improvement over any work the furnace has done in the past years. We find improvement not alone in increase of burden, but in almost every other essential feature:—viz., uniformity in quality and quantity of product, increased production, and decrease in quantity of flux—(the latter feature I had not thought of). The trial showing results so far as follows:—Increase in burden, 12; increase in production, 10; decrease in flux, from 5 to 8 per cent; and increase in quantity of No. 1 iron, 25 per cent.

"The above certainly shows a very marked improvement, and if the use of still larger charges will show a corresponding improvement, it cannot help being of great advantage. I can see but one drawback to it, and one that may prevent the increase of the charges to the size you propose in a depth of 3 to 4 feet, and that is the increased pressure. We now notice an increase of ½ to ¾ of a pound over our former average pressure; this, of course, comes from having so much fine ore in a body, and we might not have this increase, if coarser ores were used; but if the pressure should increase relatively with the size of the charges, it is a question whether we can get them up to the maximum, but a trial will settle that point. I want to give the matter a thorough test, and, therefore, make each stage of it sufficiently long to fully demonstrate the results, so that when we are through with it we can tell just the point to stop at. I shall, therefore, run a week at a time on each increased charge hereafter, and will add each time 6 barrows, so that it may be some little time yet before we can fully determine the matter. Within the next day or two, I will increase to 18 barrows, and at the end of a week note results and advise you, and will continue to do so at the end of each stage.

"Very truly,
"(Signed) GEO. D. KELLY."

"SHARPSVILLE, MERCER Co., PA., December 27, 1884.

"E. A. UEHLING, ESQ., Bethlehem, Pa.

"*Dear Sir*:—Your favor of the 17th duly received. Our No. 1 furnace is still running and doing so well that we have concluded to bank instead of blowing

out, believing the lining capable of service for a long time yet, with the exception of a few feet from the top. We are still following your method of filling with results equally as good as first advised you of. The writer owes you an apology for not writing you as promised.

"A trial of over three months fully confirms the results, and in one particular even better than we then stated, that is the diminished quantity of flux required, which will average on all kinds of mixtures from 10 to 12 per cent.

"We have not yet decided in regard to the sectional bell.

"Very truly,

"(Signed) PIERCE, KELLY & Co."

I described this method of filling in the publication mentioned below,[1] and the description was copied by several of the technical journals at the time. I have used it at all the furnaces that have come under my management, and the change from unstratified or indifferently stratified filling, to stratification of the charge, invariably resulted in a marked reduction of coke-consumption, increased production, greater regularity in running, and a higher quality of product.

The chemical reason brought forward above in favor of this method of charging, no doubt, has an appreciable share in the resulting fuel-economy; but far more important is the fact that stratification does break up the continuity of the ascending gas-currents.

The strata of coke, being so much more open than those of the ore, permit the gas-pressure to equalize itself laterally, thus preventing concentrated vertical currents which have a tendency to become more and more localized and intense, and are the major cause of nine-tenths of the irregularities occurring in the interior of the furnace.

It is evident that filling according to this method means lowering the fuel separately. The charges must be heavy enough to form distinct strata. No hard and fast rule can be laid down which would cover all conditions. The size of the furnace, physical properties of the ore, and blast-pressure available,—all are factors to be considered. The coke is lowered first; if the hopper is large enough, preferably in one charge; if not, then in two or more equal charges. The limestone should be dumped evenly around the bottom of the hopper. The ore is charged evenly over the limestone and both are lowered into the furnace together. Judgment must be used in

[1] *Stevens Institute Indicator*, 1884.

shifting the different ores composing the mixture around the hopper. Consecutive strata of ore should vary vertically as much as possible, both physically and chemically.

This method of filling, like everything human, no doubt, has its limitations. With the bell-and-hopper charging-apparatus, now in universal use, ores having an angle of repose differing greatly from that of the coke do not lend themselves well to stratified charging, since they will not cover the fuel evenly, and by producing vertically superposed annular rings of greater resistance, which may tend to concentrate the currents of gas in place of destroying them, they give unsatisfactory results. I have not had the misfortune to be called upon to manage a furnace using Mesabi ores, and cannot, therefore, speak from experience with that particular ore, but in discussing this question recently with Mr. McDonald, General Manager of the Ohio works of the Carnegie Steel Co., the above fact was first brought to my notice.

The method of heterogeneous filling described by Mr. Barrows, therefore, marks a step in advance, and will, no doubt, be welcomed by many furnacemen whose troubles have been multiplied by the introduction of the Mesabi ores. Unfortunately, this method of filling is restricted to hand-filled furnaces, thus leaving the problem still unsolved for the major part of the furnaces consuming Mesabi ore. Heterogeneous filling can evidently never be applied to automatically filled furnaces, whereas stratified filling might easily come to their rescue if proper distribution of the stock could be obtained. The only hope for relief seems, therefore, to rest on an invention that will accomplish the latter.

Standard Specifications for Pig-Iron and Iron Products.

A Discussion of the Paper by a Sub-Committee of the American Society for Testing Materials, p. 162.

ALBION S. HOWE, San Francisco, Cal. (communication to the Secretary*):—In discussing this paper, I would suggest that the dip for iron pipe be liquid asphaltum, instead of coal-tar and pitch. Asphaltum is superior to coal-tar, and the variety

* Received May 26, 1904.

which is obtained as a by-product from the California oil-wells is cheap. I have laid pipe having a coating of maltha, which was used for conveying salt water. About six years afterward, the pipe was found to be still bright, the coating not having been attacked either inside or outside. Hot asphaltum is, for this purpose, superior to the asphalt-paints made by dissolving oil-well residuum in carbon bisulphide; but it is not so convenient for retouching iron-work where a previous coating has been abraded. Asphaltum is not affected by acids, and is a good insulator. I have used it for coating cyanide-tanks, for covering iron, wood and concrete surfaces, and for rendering concrete water-tight.

In San Francisco there is a pump-room 50 ft. in diameter, with a thin floor of reinforced concrete, and an exterior wall of concrete 5 ft. thick at base and 2 ft. at top. On the sand beneath the concrete-floor there is a sheet of asphaltum roofing-felt, the joints of which were made with hot asphaltum; and the exterior of the circular wall is covered with asphaltum about $\frac{1}{8}$ in. thick. This wall stands a water-pressure of about 18 ft. head, and the interior is perfectly dry. The inside face has been painted for appearance's sake with white lead and oil.

Chemical Specifications for Pig-Iron.

Discussion of the Paper of Edgar S. Cook, p. 175.

JAMES GAYLEY, New York City (communication to the Secretary*):—The main thing that is sought after in this matter is that all purchases shall be made by analysis. This is done already in special lines, as in the manufacture of various grades of steel in which phosphorus, silicon and sulphur are specified within narrow limits, and I quite agree with Mr. Cook that specifications, which are over-rigid in unessential particulars, should be avoided, as they defeat their own purpose because unreasonable requirements permitting no variation are practically not fulfilled, and it is impossible to specify grade

* Received March 7, 1904.

and analysis in an apparatus so variable in its results as a blast-furnace.

What is desired for the consumer is an honest grade of iron, and that is obtainable far easier through stipulation as to analysis than as to grade. I do not quite understand, according to the specifications of the several grades of iron, why the committee has only specified silicon and sulphur, while the phosphorus, which has such an important influence in foundry operations, should have been omitted. The different makers produce a brand of iron which contains phosphorus varying within certain limits, but it becomes necessary for the consumer to be acquainted with the character of the brands that are on the market with respect to their content of phosphorus. It seems to me that it would be far better to stipulate by some brand-name an iron containing a certain percentage of phosphorus, as, for instance, an iron containing from 0.25 to 0.50 per cent. of phosphorus, be designated, say, as "Alpha grade;" from 0.50 to 0.75 per cent., "Beta grade;" from 0.75 to 1 per cent., "Gamma grade;" and from 1 to 1.25 per cent., "Delta grade;" etc., so that a consumer ordering iron can specify No. 1 or No. 3, Alpha or Delta grade, as will suit his requirements, and irrespective of the furnace company from which he purchases, and would be assured of obtaining an iron that is practically uniform as respects silicon, sulphur and phosphorus.

This system of grading is in use at the mines controlled by the United States Steel Corporation in the Lake Superior region, in which the name of the grade of ore beginning with A represents the best Bessemer grade; the word beginning with B represents non-Bessemer grade, and the one beginning with C represents a lower grade of ore than either of the other two, as, for instance, at the Mansfield mine the various grades are designated as Atfield, Barfield, Clearfield, and at the Buffalo mines the grades are Alamo, Buffalo, Cameo, the termination of the word being used to designate the mine in addition. This suggests to me that something on a similar basis might be used to advantage in the grading of foundry-iron.

Another point has been brought out in a discussion of iron for foundry purposes, and that is, that much trouble may arise from the content of oxygen. It occurs to me that much valuable time is lost in looking for some unusual content in pig-iron,

to which great and mysterious influences are attached, whereas the conditions under which iron is remelted, the content of sulphur in the coke, and the amount of scrap that is mixed with the iron, about which generally the founder knows little or nothing, exert a far greater influence.

I recall an experience I had when connected with the Carnegie Steel Company, in endeavoring to find out the superiority of charcoal-iron for the making of car-wheels. A certain grade of charcoal-iron, which contained about 0.75 per cent. of phosphorus, had given excellent results in the making of car-wheels, and inasmuch as we had on hand a stock of ore exceedingly low in phosphorus at one of our furnaces, we concluded to make some iron on a very low temperature of blast that would contain about 0.03 per cent. of phosphorus. This iron we made with a hot-blast temperature of 300° F., as compared with the charcoal-iron with 500° F. The resultant metal contained, approximately, as I recollect, the same percentage of silicon and carbon as the charcoal metal, but it did not yield the same results in the manufacture of car-wheels.

Before making this experiment I had consulted with Mr. Wm. Metcalf, of Pittsburg, who had long been known as a very careful investigator in iron and steel metallurgy, as to the superiority of charcoal-iron, and he had suggested that it was perhaps due to the absorption of nitrogen; that he had found a very great variation in content of nitrogen in various steels, and that the charcoal-iron with a cold blast would absorb less nitrogen than iron made with a hot blast, irrespective of whether the fuel was charcoal or anthracite. In order to test this question, the low-phosphorus iron was manufactured with a blast-temperature of 300° F. Samples of this low-phosphorus iron and the charcoal-iron, and of our standard Bessemer iron, and a piece of basic iron taken indiscriminately from a pile containing about 0.50 per cent. of phosphorus, were sent to Prof. Langley for determination of nitrogen. We looked forward with keen interest to the return from these samples, expecting that the low-phosphorus and the charcoal-iron would show so much better; but instead, the basic iron, from which we expected nothing, was materially lower in content of nitrogen, and was made perhaps with the highest temperature of hot blast. (The content of nitrogen in these

samples were: Cold blast, 0.0017; Climax, 0.0020; Bessemer, 0.0011 and Basic, 0.0015 per cent.) Since that time I have regarded with more or less indifference the claims that are put forward respecting the great influence that is exerted by the gaseous contents in pig-iron. Both the steel-maker and the foundry-man will obtain far better results when pig-iron can be made with far more uniformity than is possible under present conditions. In a furnace using Lake Superior ores it requires, approximately, 7,200 lb. of raw material and 11,700 lb. of air to produce a ton of iron. The raw material is controlled now within a variation of 10 per cent., whereas the air will vary in a single day as much as 100 per cent. Experiments are now under way which will demonstrate during the current year what increase in uniformity can be obtained in the blast-furnace by making the quantity of air consumed per ton of pig-iron as uniform as that of the raw material.

JAMES P. ROE (communication to the Secretary*):—Although Mr. Cook's paper refers chiefly to specifications of pig-iron for use in foundry work, a passing word on the use of machine-cast pig-iron and the knowledge of its chemical composition for mill-use may not be out of place.

Broadly, the greater the number of constants that can be introduced into a given operation the more uniform will be the result. Therefore, given an iron containing a known and uniform amount of the various metalloids, it is possible to obtain a product of relative uniformity; and, what is of the greatest importance, a basis for the associated ideas of cause and effect.

As an illustration of the extreme crudity of grading by fracture, the following analyses are submitted of sand-cast pig-irons shipped by the blast-furnaces as gray-forge irons and mottled irons, respectively, and similarly graded by fracture by our chemist. The analyses of these irons showed:

Gray-Forge.		Mottled.	
Sulphur.	Silicon.	Sulphur.	Silicon.
Per Cent.	Per Cent.	Per Cent.	Per Cent.
0.035	2.06	0.166	2.00
0.140	0.65	0.236	2.40
0.016	0.61	0.057	0.24

* Received March 7, 1904.

These extreme variations are by no means as uncommon as might be supposed; and they are representative, although in less degree, of sand-cast pig-iron in general. Such variations explain the wisdom of the common practice in puddling, of mixing a number of brands of pig-iron together, the probability being that only one brand would get "off" along certain lines at a given time, and that this would be neutralized wholly or in part by the others.

The advantages of machine-cast pig-iron for mill purposes are: 1, greater fusibility of a given grade, due to the absence of the relatively infusible coating of silica; 2, reduced loss, due to the same cause; and 3, relatively uniform chemical composition.

I wish to emphasize the importance of uniformity in chemical composition. The analysis of a pig that is representative of a shipment is of value, while the analysis of a pig that is not representative is valueless. A sand-cast pig is less likely to be representative than a machine-cast pig. Another feature worthy of note is that an exact knowledge of the chemical composition of a pig-iron is of more value than a too narrow limitation of the individual components. This statement is fully proved by the relatively high phosphorus and sulphur of certain brands of celebrated finished irons. In closing, I beg to call attention to an argument frequently made that, because the presence of certain metalloids in large quantities is disastrous, the presence of the same metalloids in small quantities is proportionally disastrous. This deduction is not true, for the reason that the presence of a small proportion of certain metalloids is often of positive good to the iron.

R. W. RAYMOND, New York City*:—The extracts presented in Mr. Cook's paper, from the remarks on this subject made by me a dozen years ago, represent my views at the present day so fully that I do not care to modify them. But I would emphasize, in addition, a principle involved, not only in Mr. Cook's paper, but also in many of the opinions of other experts expressed in to-day's discussion and also in previous discussions of this subject, and of others cognate to it. It is really, in my judgment, the main principle which underlies the relation be-

* Received March 7, 1904.

tween makers and users, with regard to specifications; and therefore I think it deserves to be clearly stated by itself. This principle is, that rigorous specifications, inevitably involving and inviting lax inspection, should be avoided, and that specifications should be made as liberal and lenient as is practically consistent with the purpose in view, and then enforced by an absolutely rigorous inspection.

In too many cases the contrary method has been followed. Demands have been made of the manufacturer which he could not fully satisfy. In one instance of this kind, known to me, a Government board of experts set up, for certain manufactures of steel, specifications combining the extremes of all the chemical and physical requirements contained in all the existing specifications which they could find—not realizing that the fulfilment of one such requirement would practically prevent the fulfilment of another. The result of such pedantic specifications is a two-fold mischief. In the first place, prudent manufacturers will not undertake to satisfy them; and, in the second place, adventurous or unscrupulous parties will take the job, either because they do not appreciate its difficulty, or because they expect to hoodwink the inspector, or because they rely upon some illegitimate influence to pull them through, and get them paid, in spite of the absurd conditions which they have failed to satisfy. This is, in fact, an old trick of corrupt officials, by means of which honest competitors are discouraged. I remember characterizing it, many years ago, in an article under the title, "Bulldozing Contractors."[1] But it is not by any means only corrupt officials who are guilty of it. Scientific zeal may lead honorable experts to commit the same mistake. And the cure for the resultant difficulties and mischiefs is, I think, to be found in minimum requirements of specifications, and then (but only then) rigorous enforcement thereof. In other words, let the contract cover the widest permissible variations, but give the inspector no discretionary power to enlarge them.

It is of the utmost importance that, in carrying out such a reform as is now proposed, and indeed required, by the progress of modern practice, we shall secure the means of know-

[1] *Engineering and Mining Journal*, vol. xxiii., p. 101 (February 17, 1877).

ing that we get from the maker the thing we have called for; and the only way to do this is to call for no more than we really want, and he can practically furnish.

B. F. Fackenthal, Jr., Riegelsville, Pa. (communication to the Secretary*):—The manufacturers of merchant pig-iron can congratulate themselves that foundry-men are slowly but surely recognizing the fact that their mixtures should be made up according to chemical analyses, rather than by fracture; but in my experience many foundries ordering specification-iron want the grading by fracture also. They want both the penny and the cake.

One of the largest customers of the Thomas Iron Co., who uses thousands of tons of Thomas iron yearly, always buys "No. 2 plain," stipulating for shipments from the lower pig-beds when we can furnish them. Being an old furnace-manager, he knows that the iron in the lower beds, having traveled farther from the furnace, cooling as it warmed the greater length of runner, solidifies sooner in the beds, and thus presents a closer grain; whereas, the iron of the upper beds, remaining longer liquid, will exhibit after solidification a fracture of higher grade, though the two may be identical in composition. In other words, No. 2 plain iron from the lower beds may be just as good in the cupola as No. 2X of the same cast from the upper beds, while it is cheaper, when the price is determined by grade, and the grade by fracture. This founder has no laboratory; but his practical knowledge of the conditions of the pig-bed has enabled him, for many years, to save the difference of price (at least 50 cents per ton) between No. 2 plain and No. 2X on a considerable proportion of the iron he has bought. Since classifying by analysis instead of fracture would put the No. 2 plain of lower pig-beds in the same class with the higher-priced No. 2X of the same cast and composition, it is no wonder that, as a buyer of superior shrewdness, he prefers the old method.

Yet, classifying by analysis, which is in such cases and in other ways often advantageous to the furnace-man, ought to be preferred by the founder, because it would enable him to

* Received April 23, 1904.

calculate his mixtures with certainty. For there are both tricks of nature and tricks of the trade, which may make the grading by fracture unfavorable and misleading to the consumer; and, after all, the most advantageous arrangement for both parties is that which is just to each. Unquestionably, such an arrangement should be based upon a common knowledge of the composition of the pig-iron concerned; and this knowledge requires of the founder the determination of mixtures for the cupola, based on chemical data. This, however, will be of but little value to the founder, who permits his application of chemistry to stop there and does not carry it also to the floor of his foundry, by regular tests of his castings. But some founders are not yet willing to take this trouble; and, as a consequence, they cannot locate the cause of unsatisfactory castings. Often they complain of the pig-iron, when this was entirely suitable, but was spoiled in re-melting. The following examples from my record of analyses may serve as illustrations:

1. Complaint was made that a car-load of Thomas No. 2X iron produced hard castings, when soft ones were desired, and were reasonably to be expected from that grade of pig. The customer returned two pigs, marked respectively "hard" and "soft," and two samples of the unsatisfactory hard castings. These four samples were analyzed in the Hokendauqua laboratory, with the following results:—

	Hard Pig. Per Cent.	Soft Pig. Per Cent.	Casting. Per Cent.	Casting. Per Cent.
Silicon, . . .	2.680	2.633	2.327	2.257
Sulphur, . . .	0.022	0.023	0.263	0.249

The Thomas Iron Company's sampling of the entire car-load, prior to shipment, showed silicon 2.61 and sulphur 0.022 per cent. It is unusual for the analyses of selected pigs returned to the laboratory to correspond so closely to those of the furnace sampling. The "hard" pig was doubtless chilled in the runner and, therefore, not as open-grained as the "soft" one; but they doubtless came from the same cast, and possibly from the same bed. If the founder had bought by analysis and not by fracture, he would have preferred the "hard" pig, since it was slightly higher in silicon, and slightly lower in sulphur than the "soft" one. The castings sent for analysis contained about twelve times as much sulphur as the original pig-iron,

and the founder's difficulty was certainly due to bad cupola-practice. The excessive amount of sulphur in the castings may have been due partly to fuel high in sulphur, and perhaps partly to improper fluxing, but more probably to the use of too little fuel in re-melting the iron. The loss of silicon during the re-melting was about 13.8 per cent., which shows fairly good practice in this respect.

2. Another customer, who complained that Thomas No. 2X iron gave him bad results, returned six pigs and a hard casting. The analyses of these samples, also the analysis of the pig-iron before it was shipped, were:—

	Furnace Sampling of Car-Load. Per Cent.	Six Pigs Returned. Per Cent.	Hard Casting. Per Cent.
Silicon,	2.70	2.854	2.257
Sulphur,	0.017	0.052	0.319
Phosphorus,	.....	0.812	0.800
Manganese,	.....	0.403	0.291

Loss of silicon on re-melt about 20 per cent.

In this instance the pigs returned were much higher in sulphur, but the cause of complaint was doubtless the same as in the former instance, viz.: too much sulphur was allowed to be taken up in the cupola. A few more pounds of fuel would have corrected the difficulty and kept the sulphur out of the castings. The same customer reported that a previous shipment of iron, containing silicon, 2.40; sulphur, 0.041; and phosphorus, 0.736 per cent., gave him entirely satisfactory results. I could give many examples of this kind, but these two are sufficient.

Many foundry-men call for high silicon iron; and if the results are not satisfactory, they seem to think that still higher silicon would correct the evil. Our experience, however, is that, in many instances, the silicon specified was already too high, and that they required iron lower, not higher, in this element.

3. A complaint was made of Thomas No. 2X iron, on account of the blow-holes it contained, which, the founder said, were reproduced in his castings. The iron made at that time in one of our furnaces did, in fact, contain an unusual number of blow-holes, and we experienced great difficulty in locating the trouble, which was finally traced to the quality of the sand

used for the pig-bed. The founder sent us four samples of borings, two from the pig-iron and two from the castings, which gave the following analyses :—

	Pig-Iron, Solid Part. Per Cent.	Pig-Iron, Side of Blow-Holes. Per Cent.	Castings, Solid Part. Per Cent.	Castings, Side of Blow-Holes. Per Cent.
Silicon, . . .	2.608	2.617	1.778	1.778
Sulphur, . . .	0.029	0.027	0.150	0.129
Phosphorus, . .	0.836	0.860	0.864	0.852

Loss of silicon in re-melting about 32 per cent.

I cannot see why blow-holes occasioned by bad sand in the pig-bed should be injurious to the iron, or why such blow-holes should be reproduced in the castings. Both in the pig and in the castings the sulphur was lower in the iron around the blow-holes than throughout the solid part of the pig. This is another instance, in which the iron was permitted to take up excessive sulphur in the cupola, and, doubtless, goes to show that the founder was trusting to the appearance of the pig rather than to its chemical analysis.

4. Three car-loads of pig-iron shipped by rail were delivered to the wrong customers: A puddling-furnace receiving No. 2 plain, intended for a pipe-foundry; the pipe-foundry receiving a car of No. 2X intended for a pump-works; and the pump-works receiving the car of gray-forge intended for the puddling-mill. All these shipments were received and used without complaint. In these cases the fracture could not have been taken into account by the customers.

5. To a customer operating puddling-furnaces, who always specified open iron, there was shipped by mistake a car of silver-gray iron, containing 6.38 per cent. of silicon, made while blowing-in a furnace. He reported that particular car-load as having given very satisfactory results; and for some time thereafter he referred to this shipment and wanted more like it. The iron was open-grained, and the appearance suited him. I have often wondered whether he used it alone or in mixture with some other iron that happened to be particularly low in silicon.

6. Some years ago, visiting a furnace-plant in the Birmingham district, Alabama, I found that special efforts were being made to keep the iron in the pig-beds hot for as long a time as possible, by covering it with sand to the depth of 6 or 8 in. It

was claimed that this annealing-process toughened the iron and permitted the segregation of an additional amount of graphitic carbon. On my return home, believing that I had learned something of great value, I tried the experiment at the Durham furnace, of which I was manager at that time, with the result that after the iron was cool it was impossible to break the pigs from the sows by sledge-hammers or other ordinary means. The pig-iron was finally dragged out of the cast-house with a team of horses, one bed at a time, and the entire force of blacksmiths was used to cut it apart with cold-cutters. No further attempt to improve the fracture was made.

I will close these remarks by giving the following statement of interesting experiments, made in the cupola of a large foundry, to show the losses of silicon in re-melting, as affected by the presence of manganese:—

Sample. No.	Manganese. Per Cent.	Per Cent. of Total Silicon Lost in Re-melting.
1,	0.04	34
2,	0.20	23
3,	0.43	12
4,	0.53	4

I am not at liberty to give the brands of iron, except to say that Sample No. 3 was Thomas iron.

Specifications for Cast-Iron and Finished Castings.

Discussion of the Paper by Richard Moldenke, p. 185.

RICHARD MOLDENKE, New York, N. Y. (communication to the Secretary*):—In following the discussion of the specifications for cast-iron and finished castings, I was strongly impressed with two points which might be ventilated to advantage more fully from the view of the practical foundry-man. First, the reasons why only silicon and sulphur were specified and not phosphorus, manganese and carbon; and second, why but little was said concerning the presence of iron oxide in the material.

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* Received March 23, 1904.

With regard to the first point, the modern foundry-man always demands a full analysis of the pig-iron on which he is figuring with the seller; yet, when the order is placed, only the silicon and sulphur contents of the pig-iron are of importance.

Concerning the presence of phosphorus, all the pig-iron made for foundry-purposes is divided into three classes, namely, containing less than 0.4 per cent.; between 0.4 and 0.8 per cent., and more than 0.8 per cent. In all of these three classes a choice is to be had of the silicon and sulphur contents. The pig-irons containing the lowest ranges of phosphorus will be used for making special grades of gun-iron, car-wheels and specification-castings; and to the foundry-man it is immaterial whether the phosphorus present is a mere trace or 0.4 per cent., provided it be not above the latter limit. If ordinary classes of jobbing-work, machinery, and gray-iron castings in general, pig-iron containing the medium percentage of phosphorus is desired, and in this case, also, it is not important whether the phosphorus-content be at the upper or at the lower limit, provided it does not go beyond either. Finally, for art work, stoves, and novelty castings, or those in which great fluidity of the metal is required, the high range in phosphorus is selected, and it matters little how high this range is, for the reason that any great discrepancy will be adjusted by proper mixing before melting. Furthermore, the customs of the trade are such that no pig-iron merchant would think of selling a high phosphorus pig-iron to the maker of boiler-castings, which are tested under pressure, and, therefore, might endanger life. The stove manufacturers have learned to use certain pig-irons high in phosphorus, and would not think of going outside of this class of metal; consequently, the element phosphorus is not so all-important in foundry-practice as it is in making steel; and, as a result, it was not deemed necessary to burden the general specifications for the sale of pig-iron with the phosphorus-requirements.

A similar condition of affairs exists with regard to the quantity of manganese present in pig-iron. In this case there are two classes of metal; the line of division being 0.8 per cent. of manganese. Pig-irons containing a percentage of manganese less than 0.8 per cent. are valuable for making ordinary

foundry-castings; while those pig-irons exceeding 0.8 per cent. of manganese will be sought for making special castings on account of their high manganese-content.

The founder, in glancing over the analyses presented by the agents of the various pig-iron manufacturers, will at once remove from all consideration those furnaces, which make iron too high or too low in phosphorus, or too high in manganese for his use, and his selection will be made from the rest according to the silicon and sulphur contents of the metal.

Finally, as to the carbon; if anything, the foundry-man reduces the quantity present in the pig-iron by the addition of scrap-steel in the melting-process. Therefore, but little attention is paid to the total carbon-content of American pig-irons, which is almost invariably present in proper proportion. For special castings, however, which require undue softness, a high percentage of total-carbon is sought. A pig-iron containing a carbon-content below the normal, arouses suspicion, and indicates that something has been wrong at the furnace. If an occasional specification is seen asking for a given percentage of total-carbon, it is probably necessary for a given purpose. On the other hand, a specification (and I have seen several) demanding more than a given minimum of graphite is an evidence of ignorance, and the furnace-man can only hope that, some day, the maker of a specification of this kind will learn more of foundry-metallurgy and withdraw the useless and objectionable requirement.

Taking up the second point of importance,—the oxidation of the metal,—the lengthy discussion of this subject which took place at the Atlantic City meeting showed that this expression was understood to be the absorption of oxygen from the blast, leaving it in the iron as occluded gas. This effect is not strictly that with which the foundry-man has to contend. He is troubled with an actual dissolved iron oxide, and knows that no matter what melting-process he uses, the quantity of this objectionable constituent is sure to be increased at every melting.

The steel-man is comparatively free from this trouble, because he uses much higher temperatures, and simply adds ferromanganese to remove, at once, any oxidation-products in the bath of metal. Suppose the blow of the converter were

interrupted at a point where some silicon still remained in the metal, and the metal was then cast into molds, gases would be liberated and a weak product would result.

After all, any foundry melting-process is similar to the converter-process, excepting that the blast-pressure is less, the time of action longer, and no attempt is made to refine the metal. The air-furnace with a top-blast, used in the malleable-iron process, is practically a converter with a top-blast, instead of one at the bottom or side.

It is my belief that the difference in excellence between cold-blast charcoal-iron, warm-blast charcoal-iron, and hot-blast coke-iron, all of like composition, lies solely in the degree of freedom from oxidation of the metal as it flows from the furnace-spout. Foundry-men instinctively realize this effect, and in making special castings the utmost care is taken to get pig-irons which will yield castings of the best service qualities, the composition being constant. Even in the malleable-iron industry, no one who has to make "specification-castings" (specially good work), would think of using straight Bessemer pig-iron for this purpose, even though the chemical composition meets the requirements. The selection will be "Bessemer malleable," or pig-irons with an analysis practically within the Bessemer limit for sulphur, and made of better quality, especially with the aim to avoid weakness, which really means oxidation. I have repeatedly used ordinary "Bessemer" in making malleable castings, with the result that the tensile strength was much below the average. As a result of this experience, I used the pig-iron only from those blast-furnaces, which look after these matters more closely and make a more satisfactory product.

What the foundry-man fears most, in using the open-hearth furnace for his work, is the occasional rising of pieces of the furnace-bottom, which are saturated with burnt iron, and oxidation-products. These pieces float on the surface of the bath of metal and gradually yield to it the iron oxide contained, with the result that the metal in the bath, though of proper chemical composition, loses its life as soon as tapped, and "skulls" everything into which it flows.

Foundry-men know that the pig-iron they use, be it as good as it can be made, is never improved in the melting. Even the best of them would be glad to do better work, if they knew how

to do it. The most pressing need at present is the assurance that they are getting the best pig-iron that can be made for a given composition, and that the blast-furnaces, also, are trying continually to improve their work. If both foundry-man and furnace-man will keep this improvement always in mind, there is nothing to fear for the future, and in time we may be able to correct irregularities, which are now beyond our knowledge.

Stock-Distribution and its Relation to the Life of a Blast-Furnace Lining.

A Discussion of the Paper of David Baker, p. 244.

EDWARD A. UEHLING, New York City (communication to the Secretary*):—Mr. Baker's paper is one that brings up a subject of great importance, and if full statistics could be collected of the number of furnace-linings ruined by reason of bad distribution by skip-hoists, it would be found an astonishing array. Unfortunately a universal remedy has not yet been found.

The fact that Mr. Baker has apparently succeeded in overcoming the trouble at Sydney, is no guarantee that the same remedy applied at another plant, where very different or even slightly different conditions prevail, would cure the trouble.

That it requires but a slight continuous difference of penetrability in the column of the descending stock of one section, over another, to produce channeling, is clearly demonstrated by Mr. Baker's experiments, by which he reversed the channeling from front to rear instead of remedying the trouble. Every change in the physical make-up of the charge necessarily requires a specific adjustment; even supposing this feasible, the regular working of the furnace is by no means assured, because with the high pressures now customary, the tendency of the ascending gases to plow their way through in a direct course, prevails, and having once opened up a passage, they will persist and cause trouble.

Homogeneous filling, hard driving and regularity of working are incompatible. The higher the furnace, the greater the

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* Received March 23, 1904.

troubles; and the finer the ore, the more frequent and aggravated their occurrence.

Stratified or heterogeneous fillings are the only two efficient preventatives of channeling or irregular distribution of the gas-currents that exist. Except, perhaps, in connection with the Brown charging-device, the skip-hoist cannot be used for the former, and the fine ores do not appear to lend themselves successfully to the application of the latter, in connection with the bell and hopper.

Mechanically, the skip-hoist has so many advantages that it would be hard to replace it. The Caline distributor, it is claimed, does, to a large extent, overcome the difficulty; if not, the importance of the subject is such that the proper remedy must and will be found.

DAVID BAKER, Philadelphia, Pa. (communication to the Secretary*):—Mr. Uehling is quite correct in saying that the remedy I applied to overcome the bad distribution at Sydney would not necessarily affect the same result under different conditions. In fact, there have been at Sydney, since my paper was written, developments indicating that in that case the distribution was not perfect. I am glad, however, to have directed attention to this important matter; and I hope to report later the results of observation and experiment on other methods of filling, and the means of securing uniform good distribution of stock in mechanically-filled blast-furnaces.

MR. T. F. WITHERBEE, Durango, Mexico (communication to the Secretary†):—Mr. Baker's paper is very instructive in that it shows how furnace-derangements sometimes originate from apparently slight causes.

In 1896 a similar burning through of the lining occurred at the Mayville furnace of the Northwestern Iron Co., Mayville, Wis., in exactly 38 days from the time the furnace was blown-in. At that time I attributed it to the fact that the bell was carried several inches to the front side of the furnace, the length of the versed side of the arc described by the lever, thus dumping a little more of the charge on the back side, which made the front side of the furnace-charge more open and

* April 13, 1904. † Received January 17, 1905.

consequently increased the circulation of the furnace gases through it.

After reading Mr. Baker's description, I believe that possibly a little more ore was charged on the back side of the hopper, due to the fact that it was nearest to the elevator. This unequal charging, however, was unconsciously done, as the fillers were both honest and faithful.

The illustration of the burned lining of Furnace No. 1, given in Fig. 3 of Mr. Baker's paper, answers very well for the Mayville furnace. It shows that, in both cases, the destruction of the lining was largely due to faulty construction, in that there were no cooling-plates in the angle of the bosh and in one or two courses of brick above it, where, I believe, they are more in demand than anywhere else in the furnace. The presence of these cooling-plates at the angle of the bosh would have obliged the current of the hot gases to turn the corner and would have prevented them from following the angle of the bosh completely through the lining and shell.

The Mayville furnace was hand-filled, but it always showed a tendency to work to the front; so much so, that the front tuyere, although not directly over the iron-notch, generally could not be used wide open; most of the time it was bushed down to 2 in. with a clay stopper, which served to regulate the condition of the iron-notch.

The accident at Mayville was of minor importance, a spray of water on the brick-work preventing any further damage until 12 cooling-plates could be cast in the pig-bed. A stop of 9 hours sufficed to cut 12 slots through the shell and to set the plates flush with the original inside of the lining at the level of the perforation. Thus repaired, the furnace ran 999 days, and when blown-out, the lining was found practically intact on the back side, but at the front side it had been cut through to the limit.

Without questioning Mr. Baker's diagnosis, I would suggest that his description of the behavior of his Furnace No. 1 applies also to a "cored up" furnace, or one that works very stiff in the center, which would account for the cinder and iron remaining around the tuyeres and burning them, and also for the relief which resulted from increasing the volume of the blast, as I understand it, before reducing the percentage of alumina in the slag.

Under such conditions any slag will become viscous from the cooling-effect due to the passage of the comparatively cooler blast through it, the blast never attaining the temperature of slag-fusion.

Another similar instance comes to my mind in the conditions existing at the Calumet furnace, South Chicago, in 1895, where the cinder was always "up;" break-outs high up in the boshes were of constant occurrence, and tuyeres and coolers were frequently burned out. The original dimensions were,—18 ft. by 81 ft., with a 14-ft. stock-line and an 11-ft. crucible. There were eight 7-in. tuyeres and a bell 11 ft. in diameter. This furnace had been in blast 4 or 5 months when I took charge, but I was not told that it had "bucked" ever since the campaign was started. Little by little its history leaked out.

At first, the furnace was in blast about 18 months, after which it was idle for almost the same length of time. It was then blown in with no repairs to the lining, and many of the cooling-plates projected inward a foot beyond the inner surface of brick-work, but, worst of all, the 11-ft. bell had been replaced by an 8-ft. one.

At the time I took charge, the furnace equipment was "down at the heel," 3 boilers out of 12 had been, and were constantly, off for repairs, and the largest blowing-engine was broken down;—adverse conditions which remained during three-quarters of the balance of the campaign (about three months), reducing the blast to one-half the proper volume.

The diameter of the furnace at the tuyeres had been increased to 14 ft., and, combining all of the conditions that then existed, I suspected the presence of a "core," a suspicion which was intensified by the fact that it was impossible to drive a steel bar beyond a point about 2 ft. in front of the tuyeres. By reducing the number and area of the tuyeres used, the one blowing-engine promptly removed the core.

With regard to the quantity of fine coke screened out of the bins, noted by Mr. Baker to be 4 per cent., this was about the loss on Connelsville coke delivered in Wisconsin. I have never had any experience with the Brown charging-apparatus, yet I believe it would have so distributed the materials as to avoid localizing the fine coke. At Mayville all coke was charged that would not pass through a $\frac{3}{4}$-in. mesh-screen.

In small furnaces using very coarse coke, such as Blossburg and Mexican, I have found it a decided advantage to charge dust and all, provided it was first screened and then charged regularly, in order to have a suitable resistance to the passage of the blast in the furnace. Otherwise the heat would be lost at the tuyeres, and "off" iron and other derangements would result.

I am surprised that a slag with only 15.2 per cent. of alumina should have been viscous, and it is with considerable reluctance that I give my experience with aluminous slags. I know that I shall stand almost alone, but not quite so, for there are a few furnace-men who can vouch for the utility of aluminous slags in certain cases. Three times, owing to local and commercial conditions, I have been obliged to use slags of a greater alumina-content than those which Mr. Baker found troublesome. The first time was in making foundry-iron from a rich magnetite, carrying a basic gangue, and requiring a dead flux which was supplied by adding 500 lb. of feldspar (labradorite) and 1,500 lb. of magnesian limestone.

A part analysis of the feldspar gave:

	Per Cent.	Per Cent.	Per Cent.
Silica	53.30	54.35	48.27
Alumina	22.00	36.68	15.50
Iron, magnetic oxide	7.65		14.15
Lime	5.89	.....	4.20
Titanic acid	.....	.....	8.25

The magnesian limestone had from 2 to 3 per cent. of silica, and between 13 and 14 per cent. of magnesia; the fuel was one-third Connelsville coke and two-thirds anthracite coal, and the blast-temperature was 1,400° F. A very liquid slag resulted from these conditions. The casting-flush was run into "scab-holes," and always carried a heavy back-bone of "kish," while the regular flush ran into the slag-pots of the Kloman cinder-cooler. In cooling down, both the slag-pot cakes and the "scabs" shrank away from the top crust, leaving a cavity which was studded with beautiful light olive-green, rhomboidal, plate-like crystals, super-imposed in a "staggered" order. These crystals measured from 0.25 to 0.5 in. on a side, and were from $\frac{1}{8}$ to $\frac{1}{16}$ in. thick. The fracture was smooth, showing an entire absence of porosity or grain, while the color of

the outside of the cake was grayish-green. This slag was so fluid that it ran for hours, as fast as it was made, into the slag-pots through a Luhrman cinder-notch, having only a 1.25-in. orifice. It was not in the slightest degree viscous, although it contained roughly 32 per cent. of silica and 32 per cent. of alumina.

The second case was at an all-coke furnace, using a mixture of Lake ores and a highly aluminous native ore. Slags from the foundry-iron mixture gave the following analysis:—SiO_2, 25.24; 25.32; 28.56; 30.40; 31.18; and Al_2O_3, 25.20; 24.90; 21.28; 20.24; 19.44 per cent.

Magnesian limestone was used and the slag was perfectly liquid, and the sulphur in the iron averaged about 0.02 per cent. The temperature of the blast was 1,100° F., and the fuel-consumption per ton of iron produced was 1,850 pounds.

At the same furnace all Norrie-Day ore was used for 10 days, which is known to be highly aluminous (SiO_2, 3.8, and Al_2O_3, 1.55 per cent.).

The daily slag analysis was:

Silica.	Alumina.
Per Cent	Per Cent.
27.00	20.32
30.40	19.34
21.68	28.12
21.92	28.96
22.10	28.50
26.36	25.58
28.10	24.92
29.72	25.80
28.88	26.80
31.20	25.68

The product was high-silicon, malleable Bessemer-iron, carrying also about 0.02 per cent. of sulphur. The blast-temperature was the same as on foundry-iron, and the fuel-consumption was 1,750 lb. per ton of iron produced. This was an experimental run, and there was really no call to add a siliceous ore, as was afterward done, except in deference to public opinion on slag-composition. The fuel-consumption is given to show that high-alumina caused no waste of coke.

The third case noted is the composition of the slag at Durango, Mexico, which varies from 15 to 20 per cent. of alumina

and contains low-sulphur, although the blast-temperature seldom exceeds 400° F. The slag runs through a Luhrman cinder-notch having a 1.5-in. opening, and then as far as 80 ft. on the ground. It is not viscous unless lime is short, but it is not so fluid as in the first two cases cited, which may be due either to the fact that it was actually cooler, since the blast-temperature is low, or to the fact that the flux was limestone, while in the other cases a magnesian limestone was used. I do not advance any explanation for this very abnormal result, except to suggest that a highly aluminous slag may be more fluid than one very much lower, especially if combined with magnesia. I believe that the idea of low alumina and an all-lime flux must have originated with the Bessemer furnace-men, and from their standpoint they are right, but I am sure that an aluminous slag has its advantages under certain circumstances; among others, when making foundry-iron.

Mr. N. M. Langdon said to me many years ago, when managing the Cedar Point furnace, that "high-alumina means high-silicon,"—a statement which my experience fully corroborated and which has since been confirmed by the German engineer Platz.[1] Hence, for the foundry-iron furnace-men to favor low alumina is only repeating the mistake of again rejecting the chief stone of the arch. I believe that Mr. Langdon is entitled to the credit for making this important discovery, and I would now add to it, that high-lime means open-grained iron, although occasionally somewhat at the expense of the silicon.

I have known all sorts of undesirable results to be charged to a high-alumina slag, among others that "No. 1 iron (by grain) could not be made with it;" but, at a certain furnace, that idea was at least temporarily exploded by the production of an unusually high percentage of so-called No. 1 iron.

Along in the '80s, the fad of "all-lime, low-alumina" struck the country, yet I have seen slags made on that plan with Potsdam sandstone and calcite, which would not run straight ahead the length of the cinder-spout, but would froth up and run sideways indefinitely. Mr. Geo. W. Jamme introduced the use of magnesian limestone in the Champlain district, much to our advantage, as in those days the Bessemer people

[1] Nau in *Iron Age*, January 28, 1904.

wanted about 3 per cent. of silicon and a magnesian slag did not have a sticky point, at foundry-iron temperatures. I am confident now that we were enabled to make such high-silicon iron by reason of our high-alumina, though not attributing it to that element.

What I believe to have been an equally unfounded prejudice, once existed against magnesian slags that is now attached to high-alumina, mainly on account of its supposed extra corrosive action on the brick-work, but it so happens that that part of the brick-work most exposed to corrosion—the part below the tuyeres—is the most easily and surely protected, even without the use of any cooling-plates whatever.

Alumina does not seem to be at all particular in its preference to unite with silica or lime, hence a refractory slag can be obtained by carrying alumina high, but if the same effect were sought by having an excess of lime, a "lime-set" might result.

Like Mr. Baker, I should have thought that 0.45 per cent. of titanic acid was partly responsible for the viscous slag, had I not known that Mr. C. H. Foote, while in charge of the Crown Point furnaces, once terminated a blast with ore containing 7 per cent. of titanic acid. He recently told me that he had no trouble with it whatever. I do not know how he fluxed it, but I have an idea that he simply ignored the titanic acid, and considered only the usual constituents of the ore.

When slag containing 32 per cent. of silica and 32 per cent. of alumina was made, alumina was considered a strong base, as was the custom at that time, and so by that theory it was a highly basic cinder, but if the same charge had been fluxed, taking alumina as an acid according to present practice, 2,900 lb. of limestone should have been used instead of 1,500 pounds. The iron and titanic acid, present in the labradorite was due to disseminated titaniferous iron-ore, which exists in large bodies in the formation at Lake Champlain, north of Westport, N. Y. The presence of titanic acid in the feldspar was not known until after its use as a "dead" flux, and so had nothing to do with its selection for that purpose.

I do not believe that fine ore, *per se*, is necessarily so very objectionable in the blast-furnace, but I do believe that derangements occur when it has been improperly charged, or irregularly used. Taking the case of Mesabi ores (which are

now used up to 100 per cent., according to Mr. Grammer),[2] if the charging-apparatus permits of no variation in distribution, which the conditions of the furnace may occasionally demand, it is quite natural that derangements must occur.

Ores having a varying amount of coarse and fine material should be charged proportionately with as much regularity as possible, and the same practice might apply to coarse and fine fuel, although it might spoil the record of two men charging a large furnace, by adding one or two men per shift to the labor-account; but the avoidance of a single "off cast" per week would largely repay for any extra cost.

When the Cedar Point furnace was first started, a slight irregularity was experienced on account of the varying fineness of the ore, which was delivered to the stock-pile, from drop-bottom cars on a high trestle. The coarse ore would roll to the outside of the pile, and for hours the coarse pieces formed the ore-charge; later, as the center of the ore-pile was approached, the ore-charge was made up largely of fine, and when dumped into the furnace the fine material ran down into the underlying coarse charges, locally over-burdening and under-burdening the furnace, and daily causing a cold spot and a hot spot. This bad feature was overcome by placing screens under the track and separating the coarse ore from the fine, which allowed the two kinds to be charged in relative proportions. In this case only one kind of ore was used.

Mineral Deposits of Santiago, Cuba

Discussion of the Paper of Mr. Harrison Souder, p. 308.

OLOF WENSTROM (communication to the Secretary*):—In order to do justice to a property, once the largest producer of copper in the world, which is now being reopened, with a fair promise of again becoming important, I beg to add a few words in connection with Mr. Souder's remarks on the El Cobre copper-mines.

The El Cobre copper-mines were worked by the old English

[2] See p. 136.

* Received March 1, 1904.

and Spanish companies within an area of about 1.5 by 0.25 miles, the old workings being represented by numerous open pits and more than 40 shafts, most of which were of shallow depth only and several, according to old records, were bottomed in good ore at the time the mining operations were suspended. Beyond these old surface-workings the gossan-outcroppings continue strongly for several miles easterly along the strike of the formation, The copper-bearing rock of the region is undoubtedly of volcanic nature, a tufa, usually highly decomposed near the surface, but often showing a brecciated and conglomeratic structure.

Within this rock several different vein-systems exist, the most important ones running in an E–W. direction and conforming in strike to the general trend of the volcanic rocks. The two principal E–W. vein-zones are separated by about 900 ft. of unexplored ground, on the north side of which the largest of the old mines have been opened on two or more distinct veins, whose number and continuation is difficult to trace on account of the great disturbances from faulting. Near the surface the vein-structure is entirely obliterated by the heavy decomposition of the rocks, and the ore-carrying veins are thereby replaced by an impregnation-zone of low-grade ore, in places over 200 ft. wide. The ores in lower levels are the usual copper sulphides, while nearer the surface the ore consists of copper oxides and carbonates.

Although at present the old mines are only partly unwatered, yet sufficient work has been done to indicate that valuable reserves of ore are left in the old stopes, and the exploration of the impregnated surface-croppings has resulted in exposing large quantities of ore of satisfactory richness.

During the operations of the old mines most of the ore produced in the form of lumps or concentrates containing from 15 to 25 per cent. of copper was shipped to Swansea, Wales; only a small proportion of the product was smelted at El Cobre, the smelter there not being completed until within a few years previous to the supension of work.

The reopening of these old mines assumes a significant interest in view of the importance attached to any new addition to the list of copper-producers, since, notwithstanding a strenuous search for new sources of the red metal, the discoveries

of copper-ore, however important in certain districts of this continent, have not so far kept pace with the rapidly increasing consumption of copper of the world, or the approaching depletion of many of the older copper-mines.

The Commercial Wet Lead-Assay.

A Discussion of the Paper by Mr. H. A. Guess, p. 359.

Mr. Joseph P. Gazzam, Germiston, Transvaal, So. Africa (communication to the Secretary*):—About fourteen years ago, in southeastern Missouri, I used a method for the wet determination of lead which closely resembles the modified sulphate-chromate method described by Mr. Guess. The principal difference, however, is, that after dissolving the lead chromate in hydrochloric acid, the chromic acid is determined by means of a standard solution of ferrous sulphate. The method, which I believe is still in use in southeastern Missouri, was found to be convenient and accurate for the determination of lead in the ores and various products of the mills and furnaces in that district, but it was never used for complex ores.

A number of determinations of lead in mill-tailings were made in the year 1891; a few of these results are given in Table I.

Table I.—*Determinations of Lead in Mill-Tailings by Wet Assay.*

Date. 1891.	On 16-Mesh.		On 30-Mesh.		On 120-Mesh.	
	Weight.	Lead.	Weight.	Lead.	Weight.	Lead.
	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.
May 15.....	585.0	1.03	663.5	1.19	330.5	0 89
May 16.....	511.5	1.07	651.0	1.34	374.5	0.80
May 18.....	525.0	0.98	720.0	1.10	475.0	0.74
May 19.....	514.5	1.21	731.5	1.84	468.5	1.16
May 23.....	691.0	2.05	879.5	2.30	370.5	1.44
May 25.....	500.0	1.53	712.0	2.02	397.0	0 89

Date. 1890.	Through 120-Mesh.		Total.		Original Tailings.	
	Weight.	Lead.	Weight.	Lead.	Weight.	Lead.
	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.
May 15.....	118.0	2.62	1647.0	1.200	1647.0	1.22
May 16.....	186.5	3.12	1723.5	1.335	1723.5	1.36
May 18.....	196.5	3.63	1916.5	1.237	1916.5	1.24
May 19.....	199.0	1.62	1913.5	1.481	1913.5	1.50
May 23.....	104.5	3.07	2045.5	2.099	2045.5	2.10
May 25.....	166.0	2.78	1775.0	1.700	1775.0	1.72

* Received October 15, 1904.

ARTHUR JARMAN, The University, Sydney, N. S.W., Australia (communication to the Secretary*):—I heartily endorse all that Mr. Guess says concerning the shortcomings of the molybdate titration when applied to poor lead-ores. My experience has been that, with ores containing more than 5 per cent. of lead, the results obtained by different assayers working in the ordinary way without duplicate determinations may be expected to agree within 1 per cent. of each other. If the results are important, ores containing less than 5 per cent. of lead require more care and attention than a mine-assayer can bestow on the determination.

However, accuracy is retained, if the following conditions be observed:—a "zero" or "indicator" correction should be applied, the bulk of the solution should be kept small (100cc.), the assay should be kept simmering for some time after it is apparently finished, and the solution should be re-tested before declaring it to be complete.

This claim for accuracy is clearly shown in Table II, which comprises a few results on poor ores given to students who had just completed their first acquaintance with this wet method of lead-assay.

TABLE II.—*Determination of Lead by Various Workers Using the Wet Method.*

Authority.	Sample A.	Sample B.	Sample C.	Sample D.	Sample E.
Demonstrators.	Per Cent. 8.4	Per Cent. 1.6	Per Cent. 2.4 2.1	Per Cent. 4.1	Per Cent. 4.3
Student A.	7.98 8.16				
Student B.		1.39 1.37			
Student C.			2.4 2.26 2.12	4.4 4.45	
Student D.				4.3 4.2	
Student E.				4.08 4.12	4.43 4.38
Mines-Assayer		4.5	4.5		5.5

NOTE.—Students A and B were good workers; C and E, excellent and reliable, and D, reliable but slow

* Received October 17, 1904.

The samples noted in Table II were given to the students with the instructions that 1 g. was to be used for the assay; that duplicate results were to be handed in; that the zero error was to be taken into account; and that the solution was to be kept at the simmering-point for a few minutes after being completed, and then re-tested. The only additional information given was that the lead-content was between 0 and 10 per cent.

In viewing the data in Table II, it is evident that the mine-assayer had no time to spare for such refinements of assay; otherwise, I should not have re-tested the samples before using them; similar assays, but on ores of a higher percentage of lead, being in accord with my own results. It is only with poor ores that the difficulties become a nuisance.

There are a few minor precautions which I always take in making these assays, i.e.—(1) After boiling the precipitate of lead sulphate with water, in order to dissolve out iron and other salts, place the beakers in a tray or dish of cold water for an hour in order to ensure the complete precipitation of the lead sulphate. (2) Wash the precipitate with cold dilute sulphuric acid. (3) Do not finish the washing with water, but dissolve the precipitate at once in boiling ammonium acetate solution. It is then ready for titration as usual. The ammonium acetate should contain a slight excess of ammonia in order to render the precipitate more readily soluble, and also to neutralize the trace of sulphuric acid which remained.

Without these three precautions, I do not consider it wise to assay poor lead-ores or tailings.

In order to avoid paper-slimes, remove the paper by fishing it out from the boiling acetate solution before it has had time to become disintegrated, and squeeze-out the solution that has been absorbed and held in the paper. Filter-paper, 11 cm. in diameter, carries only about 5 cc. of solution, and thus the bulk of it is recovered; then rinse the paper and fingers with water, adding the rinsings to the solution which is now ready for the titration. Except with very rich ores, the quantity of lead remaining in the paper is negligible. The bulk of the solution should be kept small by using 100 cc. of 5 E acetate solution, and never diluting it.

The formation of the precipitate is easily seen during the titration and it is noticeable that the first faint yellow colora-

tion comes on the test-plate before the precipitate ceases to form. The true end-point, a deeper yellow, appears later on, but it is not sharper than a limit of 2 drops (or 0.1 cc.) of the solution, of which 1 cc. equals 0.005 g. of lead; this means 0.05 per cent. on 1 g. of ore. I am satisfied, when certain of the end-point, that the results are accurate within 0.2 cc. of solution (4 drops), which is equvalent to 0.1 per cent. of lead in the treatment of 1 g. of ore.

The method of precipitation by chromate, and titrating the slight access, is given in Sutton's book,[1] and it is there recommended that carbon disulphide should be used as an indicator because the yellow color of the solution is said to interfere with the starch-iodide end-point.

Since reading the paper by Mr. Guess, I have made a few tests and find that the precaution mentioned by Sutton is not necessary, the end-point being perfectly good and easily recognizable, the finished solution having only a very faint green tint.

I intend to use this wet lead-method in the laboratory, since accuracy can be obtained without special precautions. Ores containing lime are not often met with in Australia, but in assaying foul slags, the lime is troublesome, and the omission of the sulphuric acid treatment should be a substantial improvement of the method.

H. A. GUESS, Cananea, Mexico (communication to the Secretary*):—With regard to Mr. Gazzam's experience with the sulphate-chromate method, used by him some years ago in southeastern Missouri, had he omitted adding the sulphuric acid in the treatment of those heavy lime ores (thereby avoiding the formation of the bulky calcium sulphate) he would have found that he could have doubled the number of assays for a given period of time without affecting the accuracy of the results. Since the date of publication of my original paper, I have ascertained by routine work that by using broad 250-c.c. beakers in this modified method, the nitric-hydrochloric solution of the ore, after adding the requisite quantity of strong ammonia and acetic acid, may be readily boiled without danger of foaming,

[1] *Volumetric Analysis*, Sutton, London (1896). * Received April 3, 1905.

and the lead may be precipitated therefrom in a thoroughly granular form by a boiling 10-per cent. solution of potassium chromate or bichromate, even if only a few milligrams of lead be present in the solution. This procedure is more expeditious and convenient than agitating the solution in flasks, as previously described and used. Further, except in the case of ores of high iron- or manganese-content, if the solution at the time of precipitation is strongly acetic, the use of 0.5 per cent. of acetic acid in the hot wash-water during the filtration may be omitted. This omission saves much time when making 50 or more analyses at once, since a jet of hot distilled water from a distilled water-stock tank can be used, and the wash-bottle is not needed until the lead chromate is ready to be dissolved in warm dilute hydrochloric acid.

Superficial Blackening and Discoloration of Rocks, Especially in Desert Regions.

A Discussion of the Paper by Professor William P. Blake, p. 371.

THEO. B. COMSTOCK, Los Angeles, Cal. (communication to the Secretary*):—Mr. Blake's recent paper upon this topic undoubtedly partly explains the rationale of a part of the known facts bearing upon discoloration and encrustation of rocks. His quotation from Humboldt, with reference to certain South American examples, may be questioned in the light of further facts, which do not appear to bear out the great traveler's theory of causation. There is no available evidence that the waters of the tropical rivers referred to by Professor Blake contain dissolved organic matter in the varying proportions required to demonstrate Humboldt's guess, nor do the concomitant facts in any manner bear out the hypothesis that the blackening in one case and its absence in the other may, perhaps, be traced to differences in the chemical reactions of the respective waters.

Careful consideration of Professor Blake's illustrations, in connection with numerous personal observations extending over widely-scattered areas, convinces me that elements of

* Received August 16, 1904.

aridity, alone, have less to do with the coloration process than the two factors of an equable temperature and a local supply of ingredients;—factors which furnish the requisites for osmosis, apparently in proportionately due degrees of intensity of alteration, as observed in both arid and humid regions. That the process itself is very slow under ordinary conditions may be inferred from the occurrence of patches of polished, blackened pebbles, cobbles, etc., in portions of the drainage-areas evidently long unsubmerged, and the absence in the same locality of similar effects even in dry stream-beds representing more recent water-courses. A third factor, air in motion, producing ozone, has also been recognized, chiefly in arid regions subjected to almost continuous currents.

I have never seen elsewhere such intense action, measured by degree of tint, thickness of alteration-zones, and brilliancy of polish, as occurs on the rocks at Cachoeira (rapids) de Almeirim, in the Tocantins river, Brazil. No possibility of wind-blown sands or other abrading agents can be accepted here, and the surfaces are frequently water-worn by the obvious action of the river-currents in flood-seasons. The zone of surface-alteration is clearly defined at a place near the average high-water mark. The water, itself, is marvelously clear, and abounds in animal life, the rocks are gneisses and metamorphic schists of Eozoic horizon, perhaps with igneous intrusions. There, the sun beats with torrid heat, vertically, but the region is humid to the greatest degree. The rocks thus affected are mainly in mid-stream, or in places well exposed to the sun, and where well shaded, the accumulation of the soil is usually thick enough to shield them.

In the western part of Wyoming, where the wind blows incessantly and powerfully, as, for instance, at Pacific Springs, near South Pass, many of the sandy layers of the Tertiary terranes are burned to a brilliant hematite-red by the action of the ozone on the ferrous oxide. But there the action appears to be wholly exogenetic. Although this region is very arid, and the proper ingredients are present without doubt, there is not the large degree of blackening effect that is characteristic in Arizona. Nor are the quartz-pebbles usually so altered, although there appears to be present every influence noted in the other cases—save one. The geographic position and alti-

tude of western Wyoming, and other climatic influences, such as the strong air-currents above mentioned, produce remarkably low temperatures at night and in winter, which more than offset the diurnal excess of heat. Although we recorded temperatures in the sand, at noon, of from 112° to 126° F., the water of the neighboring stream, carrying melted snow from the mountains, registered only 53° F. It is hardly to be expected that a change of this character will go on readily in a region where the temperatures of the air show a maximum and minimum range in one day from 115° to 32° F. The fracturing-effects of freezing, due to excessive alternate expansions and contractions, are sufficiently evident to preclude a permanent surface-alteration due to endogenetic action.

There is one other phase of this subject upon which I hesitate to touch because of insufficient evidence; but it may be worthy of more detailed study. Is it not a fact, that, in regions where conditions favor the growth of lichens (presumably feeding on similar osmotic solutions), the polished encrustations are wholly absent? This is markedly true in the Wind River mountains, Sierra Shoshone and other ranges bordering on the Yellowstone Park. In the area of wind-swept sand dunes of Sweetwater county, Wyoming, there are presented the most favorable opportunities for sand-polishing, but the polish is not there evident. Moreover, in large tracts in Arizona, the most intense polish may be observed where drifting sand is wholly out of the question.

To sum up, there appears good reason for assigning the observed effects, as does Professor Blake, to osmotic action coupled with intense heat, probably with actinic influence in fixing the coating. But I feel sure that, without fairly equable temperature (with little dependence on aridity) and the absence of mechanical soil-disturbance and due porosity in the rocks, blackening and polishing are impossible. The mineral composition of the rock, provided there be access to ferrous (perhaps more commonly manganiferous) solutions, is apparently of minor moment; for I have seen the effects intensely portrayed on a considerable variety of lithologic members, but never on any species of rock of amorphous or non-homogeneous texture.

Moreover, my observations indicate the necessity of a certain adjustment of all the factors, relatively, and this within narrow

bounds. The most important element, and probably the most limited, is porosity. Intensity of coloration cannot alone be taken as the measure of activity, nor is brilliancy of polish a criterion by itself. Perhaps this last may truly indicate the degree of actinism; but I believe that the percentage content of manganese has direct bearing, also, upon this point. Rocks of diverse composition in the same locality sometimes exhibit variations in tint and in degree of luster; *per contra*, rocks of nearly equivalent composition under diverse conditions of exposure in widely separated localities are often altered in apparent proportions to the intensity of the remaining factors. The lithologic quality of the outcrop, aside from the supply of iron and manganese, appears not to influence results materially. That is to say, one cannot possibly determine the nature of a discolored rock except by dissection. I have seen white, buff, grey, green, brown and black members, all coated jet-black, with glistening surface, and this, too, among several species in one locality.

POSTSCRIPT.—PROF. WILLIAM P. BLAKE, Tucson, Ariz. (communication to the Secretary*):—Since the publication of my paper, the Superficial Blackening and Discoloration of Rocks, Especially in Desert Regions, I have noted that similar views of the origin of superficial discoloration in the case of certain rocks from the Salt Lake basin of Utah have been expressed by Prof. George P. Merrill. In his pamphlet[1] he described the brown-coated weathered boulders from Tooele county, reaching the conclusion that the discoloration was due to the solution of the manganese and iron compounds in the interior of the boulder, while they were in the water, and the gradual bringing of this material to the surface through capillarity and its oxidation when exposed.

Improvements in the Mechanical Charging of the Modern Blast-Furnace.

A Discussion of the Paper of David Baker, p. 553.

MR. JOHN J. PORTER, Chicago Ill. (communication to the Secretary†):—Mr. Baker's account of his experiences with stock-distribution has been particularly interesting to me, as it

* Received February 28, 1905.
† Received December 16, 1904.

[1] *Bulletin U. S. Geological Survey*, No. 150.

confirms the opinions which I had formed concerning the rapid failure of linings in several furnaces that have recently come to my notice.

These furnaces, four in number, are of modern construction, having been built or remodeled within the past two years, and are equipped with a well-known type of skip-hoist and double-bell charging-apparatus. In every case, a hot spot has developed within a few months after blowing-in, which has already compelled relining in two cases, and the probabilities are strongly against a long campaign in the other two. Since the hot spot in each case appeared in a position approximately opposite to the skip-hoist, and since the sizing, mentioned by Mr. Baker, is very apparent, it would seem as though there could be no doubt concerning the cause of the trouble. Nevertheless, there are some who cannot see it in that light; for the managers of two of these plants, at least, do not admit the existence of bad distribution, and have satisfied themselves by putting in cooling-plates, thickening the lining of the in-wall, and changing the brand of the fire-brick used for the construction of the walls.

From observations of the dumping-action of the skip-hoist, and from subsequent inspection of the stock-distribution in the furnace itself, I arrived at the conclusion, some time ago, that no system of filling could be perfectly satisfactory where the stock is dumped into the hopper from one side, also that some form of rotary hopper or other circumferential distributing-device must eventually succeed the present form of apparatus. It is therefore with some satisfaction that I note the present tendency to criticize the double bell-and-hopper arrangement, and to give increasing attention to the Brown distributor.

In connection with this latter device, it has occurred to me that the construction might be somewhat simplified by substituting, for the side opening and door on the revolving-hopper, a central opening closed by a small bell, suspended in such a manner as to be free to rotate with the hopper. I have not, however, attempted to work out the details of this arrangement, but merely offer the suggestion in case any one else should care to do so.

The Concentration of Gold and Silver in Iron-Bottoms.

A Discussion of the Paper by Myrick N. Bolles, p. 666.

EDWARD KELLER, Baltimore, Md. (communication to the Secretary*):—It is pleasing to note the increasing amount of work on metallurgical problems that is being carried on by exact scientific methods, and the results of such labor issued by the laboratories of our universities. Without doubt, Mr. Bolles's paper will be welcomed by all progressive practitioners. Many of the results which Mr. Bolles presents to us are new; and I take exception to a few minor points only.

The first of his concluding remarks is as follows:

"Mattes are not homogeneous, but are composed of distinct mineral entities, and may be likened to igneous rocks, in which the mineral constituents fall out upon the lowering of the temperature and ensuing solidification."

He quotes an introductory remark to a paper[1] of mine, as follows:

"The true chemical character of copper-mattes does not yet seem to have been definitely determined, *i.e.*, it is still more or less of an open question whether they are true chemical compounds, or mixtures, such as igneous rocks."

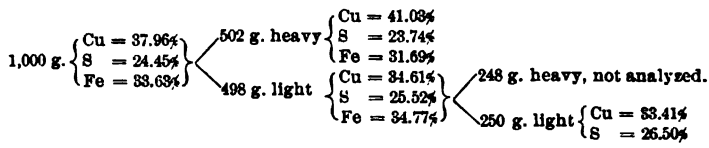
This quotation, taken alone, would convey the impression that Mr. Bolles's conclusion is entirely new. Had he quoted also another paragraph from my paper in question, it would be clear that he was, to a certain extent at least, anticipated. I take the liberty, therefore, to state the substance of my own work. In the paper referred to, I showed the existence of the phenomenon of liquation in mattes, and, continuing, said:

"A more direct method to test whether a matte is a compound sulphide or a mixture of individual sulphides was attempted by applying hydraulic separation. A clean piece of matte was selected containing practically no magnetic oxide and of the following composition: Cu, 37.96; S, 24.45; Fe, 33.63; Zn, 1.25; Pb, 1.02; Sb, 0.26; As, 0.065; insoluble, 1.15 per cent.; Ag, 26.7; Au, 0.04 oz. per ton. For such a test it is essential to select a chemically uniform piece, other-

* Received August 24, 1904.

[1] *Mineral Industry*, vol. ix., p. 240 (1900).

wise a separation of a higher- and a lower-grade matte, merely, might be obtained. The sample was all passed through a 100-mesh sieve. The separation was attempted by the primitive way of washing in a pan, and the practical end was soon reached. Starting with 1,000 g., the separation proceeded as follows:



"Although the separation does not appear as a marked success, being confined to very narrow limits, yet the results are positive, indicating individual and segregated sulphides; otherwise there could be no increase or decrease of the quantity of the several elements in the separated portions. That the insoluble portion of the original matte (a little slag) was not a disturbing factor is indicated by the increasing tenor of sulphur in the light portions.

"Although this matte, when in coarse particles, was not attracted by the magnet, yet when finely ground and under water, a separation of a small portion by magnetic means was possible. This portion was composed as follows: Cu, 31.64; S, 17.22; Fe, 46.57; Pb, 1.04; Sb, 1.25; As, 0.29 per cent.; Ag, 0.12 oz. per ton. It could not be expected to separate out by such crude means a distinct, individual substance; yet any substance separated and differing in composition from that of the matte itself, is proof of the heterogeneous character of the latter. The fact of the existence of individual sulphides established, and their extremely intimate intermixture, by the difficulty of their separation, demonstrated, it follows that the several sulphides in the molten state are mutually soluble in all proportions up to their freezing-point, at which they segregate in minute crystals or amorphous particles."

Mr. Bolles, by his excellent micrographic work, has unquestionably added valuable confirmatory data to those given in the foregoing quotation.

I would further say a few words with reference to the micrograph, Fig. 12. It shows the following structural elements: (1) the metallic iron, as black areas; (2) the ferrous sulphide, as light areas; and (3) a eutectic, as gray areas. Of the last Mr. Bolles says:

"The constitution of this eutectic is not known. Its two components deport themselves with reagents as though they were iron and sulphide respectively, but, before etching, the dark component shows a blue-gray, while the bodies of iron to which it seems allied appear as a bluish-white."

If I understand the laws of solution and equilibrium correctly, I should say that it is impossible for the eutectic to be formed of the same components which make up the larger areas of dark and light. If the latter are iron and ferrous sul-

phide respectively, and soluble in one another, then, if they form a eutectic mixture, should not one of them disappear entirely as a free structural element, and only the one present in excess be seen apart from the eutectic? The separate presence of the two components of a eutectic, together with the latter, seems to me only imaginable in the case of insufficient melting-temperature, or insufficient mixing, in the preparation of the alloy; and in such a case the experiment itself would be faulty and inconclusive.

Finally, I would say that, while the presence of metallic iron in iron-mattes may be satisfactorily established, it is certain that the condition of that element in the lower-grade copper-mattes is still undetermined. In the paper already quoted, I showed that sulphur-, copper- and iron-contents of a certain matte might be rationally written in either of the two following ways:

	Cu_2S. Per Cent.	FeS. Per Cent.	Fe_2S. Per Cent.	Fe. Per Cent.
I., ...	54.01	19.17	17.59	.....
II., ...	54.01	29.92	.....	6.84

This matte is practically non-magnetic. A matte with a tenor of magnetic oxide equivalent to this free iron, possesses, as I have shown, very marked magnetic properties. Unless the non-magnetic property of the matte in question, or rather of the metallic iron contained therein, be satisfactorily explained, it is, in the absence of chemical proof, certainly more rational to assume the presence of an iron subsulphide, Fe_2S, or an intermediary one between the latter and the common ferrous sulphide.

Regarding the identification of the constituents of his mattes, Mr. Bolles says:

"The general deportment and appearance of the constituents is sufficient to identify them, but as an additional safeguard the following tests were applied: 1. Nitric acid (2 per cent.) rapidly attacked the crystals of iron and blackened them. The ferrous sulphide was but slowly etched and the copper sulphide not at all. 2. A dilute solution of copper sulphate was applied and the iron was immediately covered with a coating of deposited copper, the ferrous sulphide being slowly coated with copper. 3. Sulphuric acid (1 : 6) etched both the iron and the ferrous sulphide, but the copper sulphide remained unattacked."

Prof. A. Ledebur has called attention[2] to the danger that, with

[2] *Trans.*, xxvii., 846 (1897).

preconceived ideas, one may readily see other than the actual constituents of the object under the microscope; and Prof. J. E. Stead,[3] highly skilled and experienced in metallography, tells us that for some time he mistook iron phosphide for iron carbide. It is therefore well to beware of appearances when determining compounds under the magnifier. Even the qualitative tests given above would probably be inadequate to distinguish positively, on the one hand, between ferrous sulphide and ferrous subsulphide, and, on the other, between ferrous subsulphide and iron. As Mr. Bolles has promised to continue his highly interesting researches, I would suggest that exact measurements of the magnetic properties might be of value in the determination of the condition of the iron in mattes.

The Application of Dry-Air Blast to the Manufacture of Iron.

Discussions of the paper of Mr. Gayley read by title at the Lake Superior Meeting, but first presented at the New York meeting of the Iron and Steel Institute, October, 1904 (see p. 746). With the exception of Mr. Pourcel's discussion (p. 1038), these papers are here published under a mutual agreement between the Councils of the two Institutes, and will also be found, with others, in the *Journal of the Iron and Steel Institute* for the year 1904, vol. ii.

Mr. E. Windsor Richards, Past-President,[1] said that in 1799 Mr. Dawson, of Low Moor, read a paper before a scientific society in York pointing out the great difference in the moisture of the air going into the blast-furnace in the hot months of the year and in the winter months. A cold blast-furnace was a very sensitive instrument, and showed this very clearly, for in June, July and August the furnace did not drive so well; it did not make the quality of iron and required more fuel; so that Mr. Gayley was on the right road to effect an enormous improvement in this direction. He considered that Mr. Gayley was on the way to great success, and hoped that the apparatus would not be a very expensive one. He supposed it was too early for Mr. Gayley to let them know what the saving was

[3] *Journ. of the Iron and Steel Institute*, ii., p. 60, 1900.

[1] Iron and Steel Institute.

likely to be. If they could produce pig-iron considerably cheaper than was done at the present time it would be a great advantage to the trade, which was at present—on the other side of the water—in a somewhat unremunerative condition. He did not know whether Mr. Gayley would be able to give some idea of the cost of this apparatus or what it would cost per ton, and what the saving was likely to be. That information would be a very great benefit to those interested in the production of pig-iron. Experiments had been made upon desiccation of air for blast-furnace purposes, but had hitherto failed owing to the enormous quantity of air to be desiccated. It seemed to him that it had been left to Mr. Gayley to solve this most difficult problem, and he congratulated him and the Institute upon the valuable paper which he had contributed.

DR. R. W. RAYMOND, Honorary Member:[2]—To my mind this paper emphasizes two propositions. The first is, that great aggregations of capital may permit investigations and experiments too expensive to be undertaken by smaller establishments, and (what is equally, if not more, important) may save the time, labor and cost which would otherwise be expended in parallel investigations, separately and even secretly conducted, each of which may fail of useful result, by reason of the lack of adequate support for conclusive tests upon a working scale.

The second proposition is, that great corporations, if they would justify their existence before the present and the coming generation, should make sure that their technical managers are both ready and able to discern the merits and encourage the development of inventions which will, by decreasing the cost or improving the quality of their products, confer lasting benefit upon the world.

Returning to the paper itself, I would call attention to what I regard as one of its most useful suggestions, namely, that the daily variation in atmospheric conditions is really more important in the running of a blast-furnace than the difference in such respects between summer and winter. In other words, we could prepare for more or less continuous conditions of any kind, even the most adverse. It would not be difficult, for in-

[2] Iron and Steel Institute.

stance, to operate a blast-furnace on the Equator, taking the blast from an atmosphere of unvarying temperature, and completely saturated with moisture. Having once determined the necessary fuel-consumption, etc., we could accept the cost thereof as inevitable, and regard the results with equanimity, as the best that could be expected under the circumstances. Similarly, if atmospheric conditions varied only with the seasons, we could prepare for expected changes and endure them with resignation. Indeed, if the variation of atmospheric moisture were confined to changes of season, it may be seriously doubted whether Mr. Gayley's apparatus would effect a sufficient saving to warrant the expense of its installation and maintenance.

But Mr. Gayley's exhibit of diurnal variations presents a consideration which, I must confess, seems to me much more important, even financially, and certainly so, when considered from the standpoint of a vigilant and anxious furnace-manager. For it was in my time, and I do not doubt that it still is, the sudden and inexplicable whims of the furnace—its changes without notice, and upon the same burden, blast and blast-temperature, which made the greatest economic trouble. We did not realize that the hygrometric variations of the air from hour to hour could account for such sudden changes; but in the light of Mr. Gayley's experiments, I think this may have been always the chief, if not the only, cause. The secret of successful furnace-management is like the secret of longevity. It is notorious that any list of centenarians will be found to represent all kinds of regimen: Early rising, late rising; total abstinence, steady drinking; tobacco, no tobacco; meat, no meat; hot baths, cold baths; sedentary indoor life, vigorous outdoor exercise, etc. The only element they possess in common is regularity and continuity of practice. That seems to be an invariable element in long life, whatever else may co-operate with it. The man who changes his habits after he reaches fifty is likely to die young! And since longevity is biologically an expression for a maximum of power and minimum cost of repairs, I think the analogy I have suggested is not fanciful, but real, and emphasizes the corresponding principle in our (otherwise totally different) department. In my judgment, the hint in this direction afforded by Mr. Gayley's paper is not its least valuable feature.

Mr. E. H. Saniter (Rotherham) said that the subject was an extremely important one, but he did not propose to deal with the main point, but rather a side-issue, as he was not a blast-furnace man himself. Incidental reference was made to the Bessemer converter. He was informed in some plants when running with only half a per cent. of silicon sufficient heat was obtained. If they took the moisture out of the air, would they be able to make metal low enough in silicon to blow at a proper temperature? Dr. Raymond hit the nail on the head when he said that what was wanted was uniformity of the air, whether it was dry or moist. It might be necessary rather to reverse Mr. Gayley's process and to add moisture by some means so as to make the air uniform, rather than to desiccate it. The only other point was with reference to its application to the open-hearth process. That might also be very useful, but they would have to treat the greater sinner—the producer-gas, because it varied more in moisture than the atmosphere. Therefore they would have to deal both with the gas and with the air. In that connection it again came to a question of uniformity. If the Mond gas, which was now used for steel-making, were taken, it would be very regular, although not very dry, and that would be a decided advantage. The drier the gas the better the heat, and in this connection the use of Mond gas was a distinct gain if sufficiently cooled.

Correspondence.

Mr. W. J. Foster (Darlaston) sent the following communication:—The question of the desiccation of that portion of the atmosphere which is necessary for the oxidation of the carbon in the hearth of the blast-furnace by means of refrigeration, will no doubt attract the special attention of blast-furnace managers in the future, especially in those places where the air required per unit of iron made is large, as compared with some furnaces which are working on more modern lines. It is quite evident that the plant that will be necessary for the desiccation of the great quantity of air required for the production of pig-iron will necessarily be of an elaborate nature, depending to a great extent on the cooling surface of the refrigerating-chamber, and the particular method of insulation, just as a boiler depends on the heating or cooling surface, assuming that the

fall in degrees of temperature in both cases are the same; but, of course, in the case of refrigerating the atmosphere the difference in the temperature of the ingoing air and the temperature of the refrigerating-chamber is not so great as would be the case when dealing with the combustible gases in boiler-practice, hence a corresponding reduction in its efficiency per unit of tube area. These conditions are based on the fact that the tubes will not conduct or neutralize the heat so rapidly with bodies at nearly the same temperature as would be the case with a great difference in the temperature of the matter inside and outside the tubes respectively. These circumstances will, of course, necessitate the erection of a plant of an elaborate nature; nevertheless, I am of opinion that the refrigerating-system is by far the best method up to the present suggested for eliminating the aqueous vapor from the atmosphere.

Previous writers on the hot-blast theory tell us that the increased coke-consumption which is due to the water or aqueous vapor entering the furnace is equivalent to the direct proportion of the heat necessary to decompose the water molecule. By referring to my paper of May, 1904, on the efficiency of the blast-furnace it will be seen that when dealing with the hot-blast question and the cooling effect of the aqueous vapor in the circumstances that were existing at the Darlaston No. 1 furnace the calculations were simply based on the heat necessary to decompose the water molecule, minus the heat which is given to the aqueous vapor due to heating the blast, only represented a saving of 0.478 cwt. of coke per ton of iron, with a total coke-consumption of 29.2 cwt.

A careful examination of my remarks on this subject will show that the above quantity of coke is not the quantity that would actually be saved, but, on the other hand, it is clearly shown that by effecting a saving in this one item by increasing the temperature of the blast we would also have a saving in every other branch of the system, with the exception of the parts clearly defined, the total of which only represents 6.646 cwt. of coke, this being a quantity that is absolutely necessary in these parts of the process which it represents, and the remaining items which correspond to 22.544 cwt. of coke would be indirectly effected by either the superheating of the blast, or any method that would have an influence on the partial re-

moval of the aqueous vapor or its decomposition before entering the furnace. The abnormally high fuel-consumption represented above is chiefly due to smelting silicates of iron with low blast-temperature, etc., and was instanced specially to illustrate the hot-blast theory: The results are very different when working with other materials.

The furnace using the largest quantity of air per unit of iron made, such as is the case in cold-blast practice, will undoubtedly have proportionately a greater advantage of the use of dry air than those using a less quantity. As previously mentioned, the saving will not be found in direct proportion to the air used, but will be found to realize very much greater advantages.

It is quite certain that the adoption of any practice that will prevent water or water vapor entering the furnace at the hearth will, without doubt, be found to have a great advantage in furnace-practice, under the conditions under which they worked at Darlaston. I have estimated that the quantity of heat units necessary to heat the blast to 454.4° C. is equivalent to 2.075 cwt. of coke (measured by the simple complete combustion of carbon and consequently based on the previously recognized hot-blast theory), and the heat necessary to decompose the water vapor was found to be 0.478 cwt. of coke, but in practice I find that this would be equal to at least 10 cwt. in the case of heating the blast to the temperature mentioned, etc., consequently the saving of fuel by removing the whole of the moisture would be represented as follows:

$$2.075 : 0.478 :: 10 :: 2.336$$

which shows a saving of 2.336 cwt. of coke per ton of pig-iron produced. It will be seen that, taking the figures mentioned above, and assuming that the whole of the moisture is eliminated, also putting the average cost of fuel at 14s., which is equivalent to 1.61 shillings per ton of iron, and allowing 20 per cent. for working-expenses and depreciation of plant, etc., estimated on 600 tons of pig-iron per week, the approximate saving in round numbers would represent a dividend of 10 per cent. per annum on a capital outlay of £20,400. From the data describing the practical results in the author's paper it appears that Mr. Gayley has solved a very difficult problem.

PROFESSOR H. M. HOWE (New York) wrote that the value of Mr. Gayley's invention was so great that one could hardly rate it justly without danger of seeming theatrical. When we considered the greatness of the pig-iron industry, even the minor advantages of the process, such as the saving of fine ore, the raising of the phosphorus-limit of the ore, and the better control over the sulphur- and silicon-contents of the pig-iron collectively, were of enormous value; but passing these by, the fuel-economy reported fairly took one's breath. Assuming that the world's annual output of pig-iron was, roughly, 46,000,000 tons, and that it called for some 46,000,000 tons of coke, representing some 66,000,000 tons of coal, Mr. Gayley's saving of 20 per cent. as shown by his Table VII. would correspond, if applied to all the furnaces of the world, to no less than 13,000,000 tons of coal per annum, or more than twice the total annual coal production of New South Wales, and more than half that of such important coal-producers as Belgium and Russia. We hardly expect this invention to be used at every furnace at home and abroad; yet when we remembered to how great a proportion of all the furnaces of the world Neilson's invention of the hot-blast had been applied, there was very little fear that any large fraction of the world's furnaces would fail to adopt the dry blast sooner or later.

His remark in the last paragraph but one, that the dry-blast plant "started without a hitch and no difficulties have been developed in any direction," would not greatly surprise those familiar with his exploits.

Refrigerating processes evidently had a great future. It was not necessary to consider such obvious cases as cooling dwellings and factories in hot climates, though failure to do this would, to the future historian, stamp this age as among the barbarous ones. But there was a use which was of interest to those who speculated as to the future of the iron industry. Not a few at this meeting had said with Burns—

> "And forward, though I canna see,
> I guess and fear."

At first sight it certainly might seem that the supply of iron-ore, finite as it was, and not reproduced like our animal and vegetable supplies, would exhaust itself, not indeed in our

time nor that of our children, but long before the other great staple materials of industry, such as wood, the textile substances animal and vegetable, breadstuffs, meat and other foods. Yet, on reflection, this idea lost its force; for if the earth really was an enormous meteor with but a relatively thin crust of rocks, there was a supply of iron which would last long after the earth had ceased to be habitable. We depended upon the forests and green fields for our supply of oxygen and of food; but when sun and earth should have so far cooled that the ice-caps spreading out from either pole finally met at the equator; when Mother Earth, exhausted, drew together those icy curtains for her endless sleep, where would be the fields and forests to give us the breath of life? Would not our atmosphere then be one of nitrogen plus carbonic acid, and the earth a desert?

But how were we to attack and reach this iron nucleus of the earth? Clearly through these refrigerating processes. When man should have exhausted the limited, and hence exhaustible, deposits of the earth's crust, knowing the vast central mass of iron beneath him, he would be forced to find a means of freezing his way to it.

Mr. V. Pendred (London) wrote: Mr. Gayley appears to have assumed, on very inadequate grounds, that moisture in the atmosphere plays an important part in the performance of a blast-furnace. How he has arrived at his conclusions I am unable to say. To simplify matters and in the end save space, I give here the only passage in the paper which sets forth the reasons which led him to make the interesting and valuable experiment which he describes. "The desiccation of the air used in blast-furnaces in such a way as to reduce its moisture to a small quantity, and to keep it uniform, must of necessity contribute in a very marked degree toward the attainment of uniformity in the furnace operations. The advantages from desiccation can be appreciated only after due consideration is given to the volume of air that is consumed per minute, and the large amount of moisture which it contains. Managers of blast-furnaces are familiar with the chilling effects produced in the hearth by a tuyere that is leaking, which immediately results in a deterioration in the grade of the iron; yet the quantity of water ordinarily entering the furnace under these con-

ditions is not greatly in excess of the quantity carried in, like a steady stream, by the atmosphere, during a period of the average humid conditions prevailing in the summer season in this country." Now I hold that although Mr. Gayley's results are satisfactory, the reasons which he gives for the success which he has attained are entirely wrong. It is evident that he has taken it for granted that water is carried into the blast-furnace with the blast. This might happen provided the blast was cold; but as a matter of fact it was heated for the Isabella furnace to somewhere about 800 degrees F., at which temperature, and some six or eight pounds above atmospheric pressure, water could not exist as such. Under normal conditions damp air is delivered to the stoves, not to the furnace. The moisture is at once converted by the red-hot bricks or pipes into superheated steam. The quantity is very small. It remains for Mr. Gayley to show what part it can play in the extremely complex reactions going on in the blast-furnace. On this aspect of the case he is entirely silent. Furthermore, it may be pointed out that the ore, as it comes from the bins, is always damp or even wet, and that the weight of water introduced in this way into the furnace is out of all proportion greater than that of the superheated steam-gas, which goes in with the blast. So far my contention is negative. That is to say, I hold that the moisture in the air could have no appreciable effect on the make of iron, either chemically or thermodynamically.

I now wish to show why the output of the Isabella Furnace was so much increased by refrigerating the air. Good was done, not by drying the air, but by augmenting its density by cooling it. Let us suppose for the sake of argument that; the pressure remaining at 14.7 pounds and the temperature 62 degrees F., one pound of air has a volume of only 6.57 cubic feet instead of the normal 13.14 cubic feet. Then, the blowing-engine remaining unaltered as to size, and running at the same speed, it will deliver per minute twice the weight of air. The work done by the engine will be the same. All the ports and passages will still be adequate. But the rate at which the furnace works depends not on the volume of air blown into it, but on the weight of oxygen passing the tuyeres every minute. Under the new conditions, everything else remaining unal-

tered, we should double the output of the blowing-engines, and, under suitable charging and tapping conditions, that of the furnace. Unfortunately, hitherto the iron-master could not increase the weight of his blast nor augment the weight of oxygen passing the tuyeres without running his blowing-engines faster. Mr. Gayley has shown us, however, how a very important advantage may be gained in this direction. Let us suppose that he was blowing, normally, at 70 degrees F., that being the inhaling temperature of the blast-tubs. Then each pound of air would have a volume of 13.342 cubic feet. If now, by refrigeration, he reduced the inhalation temperature to 40 degrees, then each pound of air could have a volume of 12.586 cubic feet only, and the weight of air sent into the stoves under the two conditions would—the blast-engine running unaltered—vary in the inverse ratio of the two pressures. In other words, the efficiency of the blowing apparatus would be increased by, say, 6 per cent. But we know that Mr. Gayley was on many occasions able to work with much larger temperature differences than these. Thus the inhalation temperature has been reduced from 82 degrees to 23 degrees, at which in round numbers the volume per pound are to each other about as 13.6 is to 12. In this case the efficiency of the blast-engines was augmented by about 13 per cent. This is the reason why Mr. Gayley had to reduce the number of revolutions. Indeed, he has recognized the truth, although he does not attach due importance to it, for he says in one part of his paper, "When the dry blast was supplied to the furnace it became necessary to reduce the revolutions of the blowing-engines, since the air supplied to the engines was lower in temperature than the natural atmosphere and contained more oxygen per cubic foot, and the tendency of the furnace was to drive too fast."[3] Nor is it to be supposed that the advantage gained was represented wholly by the percentage I have given. Any one familiar with blast-furnace working knows that an augmentation in blast density produces effects larger in proportion than could be anticipated. I have not thought it necessary to go into any detailed calculations. I hope I have said enough to make my meaning clear. The gain derived from refrigeration would have remained about the same if the water had not been re-

[3] See p. 768.

moved at all, provided, of course, that the stoves were of sufficient power. At the Isabella furnaces they appear to have been deficient, and the augmented blast density helped them on the one hand, while they were spared the loss of a little energy in converting water into superheated steam. Finally, I can assure Mr. Gayley that I have read his paper with much pleasure, and regard it as a most valuable contribution to our Journal.

MR. B. H. THWAITE (London) wrote as follows: Regarding the characteristic defects of modern blast-furnaces, the supremacy of control over the character of the operations and products of a blast-furnace has been advanced by the important success of Mr. James Gayley's enterprise and ingenuity.

It has often been claimed for the blast-furnace that as an instrument of industry it has attained a rare degree of perfection and efficiency, but in recent years it has been demonstrated that a wide margin still exists for further improvements, and especially in methods of thermal utilization. The margin has, however, been greatly reduced by the harnessing of the gas-engine to the blast-furnace. Again, the hot-blast stove efficiency is far too variable, and the contributor, along with others, is attempting to remove this element of thermal inefficiency and irregularity. An ideal blast-furnace operation would be one in which all the agents required for the production of iron are perfectly controllable in character, in application, and in proportion. But we know that the hygrometric condition of the air is constantly changing, and this variation, added to the vagaries of the thermal output of the hot-blast stoves, is responsible for many of the annoying irregularities in the working of the furnaces. Close observers of blast-furnace operation phenomena have long ago recognized, in perhaps an imperfect measure, the extent of the influence of the hygrometrical or water content variation of the air on the working character of the blast-furnace, but it remained for Mr. Gayley to measure more or less exactly the influence, and at the same time discover a practical means of regulating and reducing the proportion of water introduced in the air-blast into the furnace hearth.

In the period of inception of his refrigerating air-drying system, Mr. Gayley would no doubt be told that the loss of ther-

mal energy involved in his system would probably more than counterbalance the advantages to be derived in reducing to a fixed degree the moisture in the atmospheric air required for furnace-operations. To his great credit, Mr. Gayley has had the courage to balance the thermal loss and gain by an actual test on a large and expensive scale, and the balance appears to have splendily justified the test.

The introduction of water into the furnace hearth, whether in the form of aqueous vapor or directly from tuyere leakage, has the following chemical sequence when in contact with iron. The equation is $4H_2O + 3Fe = Fe_3O_4 + 8H$; in contact with the carbon of the fuel the equation is $H_2O + C = CO + 2H$.

The absorption of thermal energy in the dissociation of H_2O reduces the furnace-hearth temperature; and, although the re-oxidation of the hydrogen in the hot-blast stoves and in the steam-boiler furnaces restores part of this thermal loss, there is always in the use of hydrogen the irrecoverable loss of the latent heat of steam-formation—unless methods of condensation, or temperature-reduction below 212° F., are available, which for stove- and furnace-gases they are not.

According to Mr. Gayley, the operation of the refrigerating-machinery involves the expenditure of 535 indicated horse-power.

The writer demonstrated, many years ago, that if the blast-furnace gas was used directly for the production of the power, the output would be increased some 400 per cent.; but, assuming only a factor of 300 per cent., there would be an excess of power available by the displacement of the three steam blowing-engines of 671 indicated horse power of not less than $3 \times 671 = 2{,}013$ indicated horse power; so that the question of additional power-supply need not influence the decision as to the wisdom or otherwise of adopting the dry-air process, once the complete adoption of gas-power is also agreed upon.

In the power-expenditure data supplied by Mr. Gayley there is perhaps some miscalculation; the mystery is, however, partly cleared up by the reduction in the resistance of the furnace-burden and the reduced volume of air—represented by the fall of the blast-pressure by 8.8 per cent.

The calorific value of the blast-furnace gas of 22.3 per cent. of CO in using dry air is reduced by 2.4 per cent. or to 19.9 per cent. in the absence of the hydrogen-data; the calorific

value of the gas cannot be determined, but it will probably not exceed 75 to 80 B.t.u. per cubic foot. This gas, both for stoves lighting from the cold or for steam raising, will be very deficient in igniting qualities, and it would be interesting to hear whether this gas is used for steam raising, and whether any auxiliary fuel is required to sustain ignition. The gas is quite suitable for use in specially designed gas-engines, and its calorific reduction could be still further extended without risk of destroying its power-producing usefulness.

Advantages and Disadvantages of the Dry-Air System.

Debit.	Credit.	Credit Balance.
Thermal loss in compressing air.	Greater density of refrigerated air involves less blowing-engine power to provide same weight of air-blast.	Twenty per cent. greater output of iron.
Ditto in lowering temperature of air some 53° F.	Less resistance to flow of gases through furnace burden, owing to more equable and uniform working conditions.	Twenty per cent. reduction in fuel consumed per unit of pig-iron and output.
Reduced calorific value of gas.	Greater control over furnace-operations and its product, especially in proportion of silicon.	
Reduced sensitiveness of gas to ignition.	Reduced iron-ore waste.	Reduced iron-ore waste and greater control over silicon range.
Capital expenditure in refrigerating-plant and cost of supervision and stoves relating thereto.		

On the basis of Mr. Gayley's figures representing the economy to be secured by the use of dry air, and assuming the variation of moisture in the atmosphere between the winter and summer months to be within the limits indicated in the graphic diagram, Fig. 10, it is possible to calculate the economy that should be obtained in a furnace, released from an environment of steam issuing from blowing-engine pump and hoisting-engine and shunting-locomotive exhaust-pipes, and the escaping steam from steam-boiler safety-valves, all changes resulting from the supersession of steam by gas-power. It may also be assumed that the tuyere-water is cooled, without adding to the air moisture-content, and that the chilling of the pigs does not involve any material production of steam or water-

vapors. Taking the average proportion of moisture in the winter months of November, December, January and February to be equal to 2.15 grains per cubic foot, this proportion, compared with that of summer months' average, is only 73 per cent. less than the average content of refrigerated air in Mr. Gayley's reported tests. Now the average increase in pig-iron output as the result of using dry air equals 20 per cent., and, curiously enough, the reduction in fuel (coke) is represented by the same figure. Therefore, furnaces working in the four winter months named, and in a completely steamless environment, and all other conditions being equal, should give an increased efficiency represented by 14.6 per cent. In other words, the pig-iron output in November, December, January and February should be 14.6 greater, and the fuel-consumption per ton of pig-iron output should be 14.6 per cent. less than the results of the four summer months of June, July, August and September. There is in these possible winter improvements in output and fuel-economy an adequate *raison d'etre* for the supersession of steam by gas-power, a technical reform that would add to the advantages claimed for the dry-air blast, because—

1st. The evils of escaping steam into the atmosphere would be removed; and

2nd. There would be an ample supply of power available for air-compression and air-refrigerator services.

MR. GAYLEY, in reply, said: With reference to one question that had been asked, as to the changing of the stoves, he would state that there had been no change in their operation. The furnace at the Isabella Works, having four stoves to the furnace, was to have one stove on blast and three on gas, changing stoves every hour. Mr. Saniter had said that he understood, with respect to their Bessemer practice, that his firm was hoping to run with 0.5 per cent. of silicon; no doubt this could be done if the dry blast were used at the Bessemer department, and by such use a further economy in coke could be obtained at the blast-furnace, but in their practice at the present time, with the use of mixer-metal, the silicon-content was from 1 to 1.25 per cent. Mr. Saniter was quite correct in calling attention to the importance to the open-hearth furnace of extracting the moisture from the producer-gas. He (Mr. Gayley) was confident that if the moisture were extracted from both gas and air

for use in the open-hearth furnace, and from the air supplied to the converter, it would result in increased product and greater uniformity in the metal. A question had been asked by Mr. Windsor Richards as to the saving obtained by this process. That, of course, varied with each locality, but he had given data from which the economy for any one district could be calculated. It had been found, as a result of several tests, that the saving in horse power in the blowing-engine room practically compensated for the expense of operating the dry-blast plant. They had found through the use of this process a saving in coke, labor, supplies, and general expenses, in the limestone corresponding to the coke saved, which would otherwise have been needed for fluxing the ash, a reduction of phosphorus in the metal corresponding to the reduction in coke-consumption, which was of some importance in the making of Bessemer pig, and, where fine ores are used, there was less waste of ore through furnace-gases. In addition to this there was increased earning capacity for the plant through increased output.

In preparing this paper he had only been able, by reason of the limited time permitted in its preparation, to present a description of the process with the results obtained therefrom, while there were many questions relating to the economy effected which would have to be considered later. In the meanwhile it had transpired that certain statements made in the paper had been misunderstood, and he took this opportunity to shed further light, if possible, on some of them. First, with regard to the density of the blast; some erroneous calculations might be made from the data showing the air entering the refrigerating-chamber at about 80 degrees and leaving it at about 25 degrees, since the dry air in passing through the main from the refrigerator to the blowing-engines was increased in temperature to 35 degrees F. From a careful indicating of the engines it had been found that there was 8.2 per cent. less weight of air entering the furnace than had been used prior to the introduction of the dry blast. Next, with regard to the moisture contained in the atmosphere. It had been assumed to be a matter of common knowledge that the moisture in the air was contained as aqueous vapor, and when passed through the stoves became superheated steam, and whether the moisture entering

the furnace as superheated steam with the blast, or as water from a leaking tuyere which was at once converted into superheated steam, it was nevertheless a form of water, and that it must be dissociated by the fuel and heat absorbed; but inasmuch as there was superheated steam to deal with at one end of the process, and frost or snow at the other end, it had been deemed preferable, as stated in the beginning of the paper, to represent the moisture contained in the atmosphere as grains of water per cubic foot of air.

In the paper presented to the Institute he had simply given the results obtained by the use of dry air, and without any intention of claiming that the economy in fuel was represented entirely by the weight of moisture eliminated. It would require a very simple calculation to show that it was not. The calculations covering a period of operation showed that the saving, directly due to the removal of moisture, represented 24 per cent. of the total fuel saved. But a far greater economy was effected in the use of the dry blast, and was represented by the reduction in temperature of the escaping gases and the greater efficiency in the reducing gases as shown by the increase in percentage of carbonic anhydride. These items alone would account for one-half of the saving in coke. In addition to the items already mentioned there was a material saving through the increased temperature of the hot blast, a lowering of silicon in the metal, and in a smaller amount of slag to be melted, as less coke was required per ton of iron.

The moisture charged into the furnace through the raw material was many times greater than would enter the furnace from the atmosphere; but to get rid of it through simple evaporation by gases, which had already performed their useful work, was quite a different proposition from that of dissociating the quantity of moisture carried in from the atmosphere into the hearth of the furnace where temperature was such a vital element.

During the month of December there had been an excellent opportunity of noting the results obtained by the use of dry-blast, as Nos. 1 and 3 furnaces at the Isabella plant (No. 2 furnace having been out of blast) were making the same grade of iron. These furnaces were nearly the same size, No. 3 being a little larger. The same ore-mixture and fuel were used in each

furnace, and the conditions under which they were operated were identical. No. 1 furnace was blown with dry blast containing 1 grain of moisture per cubic foot of air, while No. 3 furnace used natural air, which, being drawn from the engine-room, contained from 2 to 4 grains of moisture. Although this period represented the best conditions as to dryness of the air in that climate, and furnaces under ordinary conditions increased decidedly in output above the average, yet the results obtained from a small and uniform content of moisture in the dried air supplied to the blowing-engines, in contrast with the varying content in the atmosphere, were very striking. From December 1 to 22, inclusive, the results were as follows:

	No. 1 Furnace. Dry Blast.	No. 3 Furnace.
Average daily product, . . .	449 tons.	400 tons.
Average coke-consumption, . .	1,858 lb.	2,309 lb.

As connections had been made to the dry-blast main for four blowing-engines, and as only three had been used on No. 1 furnace, it was decided to connect the fourth engine to the main, and supply No. 3 furnace with one-third dry blast. This was done on December 23, and it was decided not to make any increase in the burden or to change the revolutions of the blowing-engines, in order to see what effect the use of approximately one-third dry air would have on the output of iron. The furnace immediately began to drive faster, and the average daily product from December 23 to 31, inclusive, was 461 tons, an increase of 61 tons per day.

The increased efficiency of the furnace could be directed, in the main, to either increased output or to economy of fuel, according to the location of the works and as the commercial problem suggested.

The drying of the blast found, to some extent, its equivalent in an increase of temperature in the hot blast which was wholly available, as the removal of the moisture represented a calorific gain in the furnace-hearth proportionate to the quantity of moisture removed; but the dry blast provided uniformity, which the hot blast did not provide.

ALEXANDRE POURCEL, Paris, France (communication to the Secretary, received January 30, 1905)[4]:—It seems *a priori* highly

[4] Translated from *Bulletin de la Société d'Encouragement pour l'Industrie Nationale*, vol. cvii., No. 1, January 31, 1905, p. 29, of the section, La Revue de Metallurgie.

probable, in fact almost certain, as Mr. Lodin has said in his communication to the Academy of Sciences,[5] that the economy of fuel and motive power per ton of pig-iron which might be realized in Europe from Mr. Gayley's invention would not offset the cost of the installation and maintenance of the machinery and apparatus necessary for the desiccation of the blast.

There are on the Continent, even in France, blast-furnaces which consume only 950 kg. of coke per metric ton of pig, with a charge yielding but 33 to 34 per cent. of iron (instead of 43.5 per cent., as at the Isabella furnace), and a fuel containing 15 per cent. of ash and 5 per cent. of water (instead of 11.5 per cent. of ash, like the coke used by Mr. Gayley, who does not give the percentage of water).

It appears almost certain that with a charge producing 43.5 per cent., these furnaces would reduce their consumption of coke per ton to 850 or even 820 kilograms.

It is understood that the main object at these furnaces is the production, for the open-hearth process, of a basic pig-iron, containing less than 1 per cent. of Si, with from 1.5 to 2 per cent. of Mn, from 0.05 to 0.06 of S, and 1.80 of phosphorus.

But the blast at these furnaces is heated to 700° C. and more (instead of from 375° to 400°); moreover, the internal size of the stack varies from 2.5 to 3 cu. m. per ton of daily product; while at the Isabella, with an internal size of 512 cu. m., Mr. Gayley secured from moist blast a daily product of 358 tons, which was raised to 447 tons with the dry blast.

Mr. Gayley, we must remember, has been the promoter of rapid steps of progress. His communication on blast-furnace practice to the New York meeting in 1890 of the Iron and Steel Institute made a sensation. The type of blast-furnace which he recommended (that of the Edgar Thomson Works) had a height of 27 m. and a capacity of 550 cu. m., and, for a production of 350 tons in 24 hours of acid Bessemer pig, the consumption of coke per ton varied from 775 to 840 kg. per ton. The ore yielded 62 per cent. of iron, and the coke contained 10 per cent. of ash. The temperature of the blast varied from 600° to 625° C., and the volume was 730 cu. m. per minute.

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[5] *Genie Civil*, December 10, 1904.

With rich and highly reducible ores, like those of Lake Superior, especially the soft hematites of Michigan, which are almost identical with the old La Vena ore of Bilbao (now exhausted), there is no inconvenience in rapid running—quite the contrary. But this is not the case with the ores of the East and of Luxembourg, as experience has proved.

The production from these ores in a furnace of from 500 to 550 cu. m. capacity seldom exceeded 150 tons in 24 hours at the normal rate of running for "Thomas" pig-iron; and if the temperature of the blast at these furnaces is kept as regularly as possible, above 700° C., the pressure established in the blowing-engines rarely reaches 40 cm. of mercury.

Under conditions of operation so widely different from those of the Isabella blast-furnaces, there might well be a loss of economy of motive power, instead of a gain more proportionally approaching that of 136 h.p., which Mr. Gayley says he has realized.

In any case, the fuel-economy realized at the Isabella may have many explanations. As Mr. H. Le Chatelier has very courteously said, "we cannot doubt the exactness of the facts stated by Mr. Gayley," and the scientific explanation which he gives has a very real value; nevertheless, the separation of aqueous vapor from the moist blast appears also to deserve to be taken into account.

Mr. Gayley did not push the production from 358 to 447 tons until regularity of working appeared to him well established by the uniform descent of the charge, the absence of almost periodical slips of more or less voluminous agglomerated masses of half-reduced material, separated from the charge, the diminution and the uniformity of the temperature of the tunnel-head gases, the confinement within normal limits of the variation of blast-temperature between two changes of apparatus, etc. His increase of production was obtained with a smaller quantity of blast per minute—960, instead of 1,130 cubic meters.

That his hot-blast apparatus was able to maintain the temperature at 465° instead of 375° C. is explained: (1) By the smaller weight of the blast; (2) By the decrease of aqueous vapor contained in the dried air, the specific heat of steam being more than double that of air, and proportionally affecting

the process of raising the temperature in the stoves; (3) By the corresponding decrease in the proportion of aqueous vapor contained in the tunnel-head gases. These gases, though poorer in CO (20, in place of 22.3 per cent.), have, by reason of their smaller proportion of aqueous vapor, a higher temperature of combustion.

The heat developed at the tuyere-level would be increased in notable proportion, in order that the same quantity of coke per charge may be able to support the increase of ore and flux indicated by Mr. Gayley, without marked alteration in the composition of the iron obtained, apart from the very certain decrease of manganese. In fact, as Mr. H. Le Chatelier has convincingly demonstrated, the desulphurization of the metal, as a result of the elimination of aqueous vapor, from the blast, permits a lower degree of basicity, and consequent higher fusibility, with lower specific heat (nearer 400 than 500 calories), of the slag; but the proportion of manganese incorporated in the metal is thus diminished. Nevertheless, this is not an objection in a Martin-Siemens basic pig, so long as it has the essential qualifications of low-silicon and sulphur. In Luxembourg and in Germany, such an iron is designated by the mark "M. O."; that is to say, without manganese.

It is almost evident, without further argument, that all this combination of circumstances should tend to increase very sensibly, through the use of dry air, the quantity of available calories in the crucible. For the desiccation of the blast does away with the loss of heat at the tuyeres, due to the dissociation of the aqueous vapor of the blast, and also with the subsequent rise in temperature resulting from its re-constitution ($CO_2 + H_2 = H_2O + CO$) in the neighboring zone of the bosh, which renders viscous the fusible matter in the charge, forming an agglomerate which adheres to the walls. The periodical fall into the crucible of portions of these pasty half-reduced masses, when the accumulations which they form on the walls become too heavy to adhere, chills the zone below, and, consequently, debases more or less the quality of the pig-iron.

In other words, the temperature is lowered in the zone where its rise would have the pernicious effect of rendering the charge soft, pasty and tight, while, on the other hand, the zone in which carbonic oxide is liable to be decomposed into $C + CO_2$,

so as to cement the ore, is extended. The diffusion of C in the ore, by the decomposition of CO, begins below 200° and reaches its greatest activity at 450°, terminating at 800° C. This kind of cementation is above all the most important coefficient economy of fuel in the reduction of the ore. The more we prolong this reaction by enlarging the zone in which it is produced, the more we can diminish the proportion of CO in the tunnel-head gases at the mouth of the furnace, and, consequently, lower the proportion of coke consumed per ton of iron.

It is then not without reason that we place the effects of the dissociation of aqueous vapor among the causes which might explain the economy shown at the Isabella furnace as a result of the use of dry air.

Mr. Gayley has noted also, as a consequence of the desiccation of the blast, a decrease in the amount of phosphorus in the iron produced, and a decrease in the loss of fine ore, carried out of the furnace by the gases, from 5 to 1 per cent.

The first fact may be explained only through the saving in coke, if the ash from it is high in phosphorus (which is not stated). The second fact should be due to the lowering of the gases at the mouth of the furnace, resulting from the acknowledged lowering of their temperature from 281° to 191° C.

It is always dangerous to make predictions; and, in the present case, one would run the risk of being a bad prophet in predicting that the great manufacturers of iron on our Continent will never find an immediate profit by putting in practice the idea of Mr. Gayley. The idea in itself is correct: Drying the air blown into a blast-furnace may assure regularity of running, and more or less economy of fuel; but it may remain to discover a method of realizing this end less complex, and, above all, less onerous than the one by the proposal of which Mr. Gayley has distinguished himself.

The Influence of Carbon, Phosphorus, Manganese and Sulphur on the Tensile Strength of Open-Hearth Steel.

A discussion of the paper by Mr. Campbell, which was read by title at the Lake Superior meeting, but first presented at the New York meeting of the Iron and Steel Institute, October, 1904 (see p. 772). These papers are here published under a mutual agreement between the Councils of the two Institutes, and will also be found in the *Journal of the Iron and Steel Institute* for the year 1904, vol. ii.

MR. WILLIAM R. WEBSTER, Philadelphia, Pa., said: We are under great obligation to Mr. Campbell for his exhaustive investigation, and the results he has put before us to-day. They are the most complete series of tests that have ever been made, where all the conditions of rolling, etc., were kept uniform.

In discussing Mr. Campbell's first investigations, I claimed first, that the method of least squares, then used, would of necessity only give the average value of any element, and that if the effect of any element was greater in the presence of high carbon than with low carbon this would not be shown by the method of least squares. Secondly, that the effect of manganese should be considered on acid steel as well as on basic steel, instead of ignoring all the effect of that element when it was below 0.60 per cent., as was then recommended by Mr. Campbell. I am pleased to note that the results of his present investigation confirm my views on these points.

In presenting the results of his first investigation, Mr. Campbell referred to my investigation of 1893–94, and took exception to the value of phosphorus varying (depending on the amount of carbon present) from 800 to 1,500 lb. for each 0.01 per cent. He claimed that one might just as well consider the effect of phosphorus as being constant, and of carbon varying. I refer to this, as he now has introduced very similar conditions in his present tables, in that he gives variable values for manganese (depending on the amount of carbon present) for both acid and basic steels. Can we now say, with more certainty than formerly, which element should be considered as having a constant effect?

There is one other point that I took up in my former discus-

sions, it is the matter of grouping the tests together, and getting an average of each element in the steels of the group—and from these average results working out the values. I worked with the individual tests and analyses, and consider them much better. I trust Mr. Campbell will give us the full results for each individual test, in order that others may be able to work out the values for each element by their methods. This, no doubt, would result in bringing into line the views of the different investigators on this important subject. For instance, would not the individual tests and analyses assist in deciding whether phosphorus should vary from one to one and seven-eighths times the effect of carbon, as I have it in my paper before this Institute in 1894? Or should the effect of phosphorus be constant and that of manganese vary in accordance with the amount of carbon present, as Mr. Campbell has it in his present paper? But, of course, it is much better to keep the effect of carbon per unit constant under all conditions, if possible, as that is the element we depend on the most.

Mr. Campbell's present investigation will greatly assist in harmonizing the work of others, as they have all worked with the method of "successive approximations," that is, "cutting and trying," and it will now be a very simple matter to compare results. In my own case I have no disposition to insist on the former value given for each element, which, though based upon numerous careful observations and proved by many subsequent tests to be approximately reliable in practice, are still open for correction, and will be unhesitatingly withdrawn whenever any other shall be shown to fit the observed facts more closely.

The practical value of the estimated ultimate strengths is now recognized by the steel-manufacturers, and they use them in their everyday work. The color carbon determinations are used on account of the quickness of the method. I have not had time to study Mr. Campbell's paper as it deserves, but I have attempted to put his results in convenient form for comparison with those of Mr. Cunningham and my own. A table prepared in 1902 gives the values for each 0.01 per cent. of phosphorus up to 0.08 per cent., in connection with each five points of carbon from 0.06 per cent. to 0.60 per cent., and each five points of manganese from 0.20 per cent. to 0.60 per cent.

Using Mr. Campbell's new values for carbon, phosphorus and manganese for acid and basic steels, and the carbon by both color and combustion, a new table was prepared, and a comparison of the results shows that, notwithstanding the differences of the values of each element used by the different investigators, the estimated ultimate strength by the different methods agree much more closely than we would expect. It would therefore seem to me that this matter will yet be very much simplified. For instance, have we enough data before us to say that the same amount of carbon in acid steel has from 30 to 40 per cent. greater effect than in basic steel? In the present case this greater value for carbon in acid steel is compensated for by giving the manganese a much greater value in basic steel, and also by using a higher value for pure iron in basic steel than in acid steel.

The factor "R," given in all equations as a variable to allow for heat-treatment, is of the most importance, as it has considerable effect on the physical properties of the steel. Formerly it was much neglected, but since the mills have been grading the steel by the estimated ultimate strengths, the heat-treatment in rolling has been much more closely watched. In 1894, I made a strong plea for an investigation to be made on the heat-treatment of steel in connection with the work of rolling and forging. A great deal has been done in this line since then, but there is much still to be done.

I agree with Mr. Campbell in that it is not necessary to take the microstructure of the steel into consideration from the standpoint of his investigation. Yet anything that will in any way assist in controlling the heat-treatment of the steel should be looked into. In the ordinary microscopical work they have not tied up the fractures of nicked and broken pieces of steel, as we know it with the microstructure of the same steel. This step from the old to the new has long been needed, and I now desire to call attention to a method of slight etching and low magnification with a hand-glass that is very promising. It looks as though by this method we will be able to tie up the fractures of steel of, say, 0.50 carbon and under. These pieces are from the same bar of 0.35 carbon acid open-hearth steel. One piece was overheated and shows the large coarse structure; the other piece was overheated and then annealed; it shows a

much finer structure. A test of this kind would be useful in the case of large driving-axles, as a small spot could be polished and etched; the glass would show if the steel had been finished at too high a temperature in forging; that is, if it had too large a grain and in a dangerous condition. This method of investigation is offered at this time as a suggestion, in the hopes that others will take it up and improve on it.

DR. CHARLES B. DUDLEY (Altoona, Pa.) said: I would like to say, with regard to Mr. Campbell's paper, that it has been known for years, I think, that there is a relation between the chemistry and the physical properties of a piece of steel. In the course of our work in connection with the Pennsylvania railroad, we not infrequently have to examine broken parts. We do so by making both physical test and chemical analysis, and for a long time we have been impressed with a close relation between the chemical analysis and the physical properties. We ourselves, however, have never made any attempt to put this relation into figures. We can but admire the enormous amount of work that has been put upon this subject by Mr. Webster and Mr. Cunningham, and last and perhaps best of all by Mr. Campbell.

There is one phase of the question which Mr. Campbell does not claim for his work, and which we find very useful, namely, at times it is not possible to get enough of a broken part to make a tensile test. In that case we make an analysis, and interpret that analysis into a tensile test. We are constantly doing this, and are willing to say that the information so obtained is of great value. Furthermore, as is well known, most of the specifications of the Pennsylvania railroad are both physical and chemical, and in making those specifications it is necessary that the chemistry and physical properties should agree, and here again is another place where we use the figures obtained by Mr. Campbell. To my mind his work is worthy of all praise.

MR. ROBERT W. HUNT (Chicago) said: It gives me great pleasure to add my tribute in praise of Mr. Campbell's work. In the old days I do not know that I should have been so glad to have had the definite formula for steel presented, with the definite order as to exactly how the process of manufacture should proceed, the chemistry prescribed, and certain physical

results demanded. But now that I am on the other side of the house, I think it well that these prescriptions should be made. The great point is that they should be intelligently given out; that is, based upon such information as to-day intelligent research presents to you. We all, who have been connected with the steel industry for many years, know that for a long time steel was regarded as a sort of unknown problem, and a hopeless one, so that its idiosyncrasies should be accepted with regret, but still accepted. But that time is passing away. Now we can pretty nearly always tell why those peculiarities occur. I have no doubt as the investigations proceed, the time will happily come when we can always tell. Dr. Dudley has done more than any other expert in this country to draw correct deductions from the points that he mentioned to-day; that is, from the failures. In some of the discussions of the American Society of Civil Engineers the value of the scrap-pile has been enlarged upon; notably by one of its past-presidents; and that scrap-pile is a lexicon from which all manufacturers of steel can derive valuable knowledge.

Dr. J. A. Mathews (Syracuse) said that his experience had been entirely in other grades of steel than those which Mr. Campbell had brought forward, so he did not think he could add anything of value. The relation between the manganese and the sulphur might be considered specially, inasmuch as it was generally considered that the manganese and the sulphur combined chemically so far as there was an excess of manganese, and then the free manganese uncombined might exert an effect independent of the total manganese. That was merely one phase that occurred to him as being worthy of looking into. He was sorry that his experience had not taken him into this field. He appreciated very greatly what Mr. Campbell had done, and thought it would be of immense value to the industry upon the structure of that grade of steel which is so important.

Mr. Campbell, in reply, said: Mr. Webster raised the question whether the lower value for carbon in basic steel was not due to the higher value of the base. A little calculation would show that this criticism applied only to a limited range of low carbon steels. The value of carbon in acid steel was 1,000 and for basic steel 770 pounds. In a steel of 0.30 per cent. carbon the effect of carbon in the one case would be 30,000 lb. and in

the second 23,100, a difference of nearly 7,000 lb., while the base for basic steel is only 1,500 lb. higher than for acid metal. It is quite evident that no change in the value of the base can make up for an actual difference in the effect of carbon. A base can arbitrarily be assumed, and a value found for carbon for low steels, or another can be assumed and another value found for high steels, but if a formula is to fit both high and low steels, the base cannot be assumed, but must be worked out from the records. It was also suggested that a list of the individual heats should be printed, but it seemed unnecessary to burden the pages of the *Journal* with the chemical and physical records of 1,800 heats. The full record, however, would be given to Mr. Webster for his investigations.

A cordial vote of thanks to Mr. Campbell was passed unanimously.

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447	9	For "carbonic dioxide" read "carbon dioxide."
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856	9	For "from" read "form."
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